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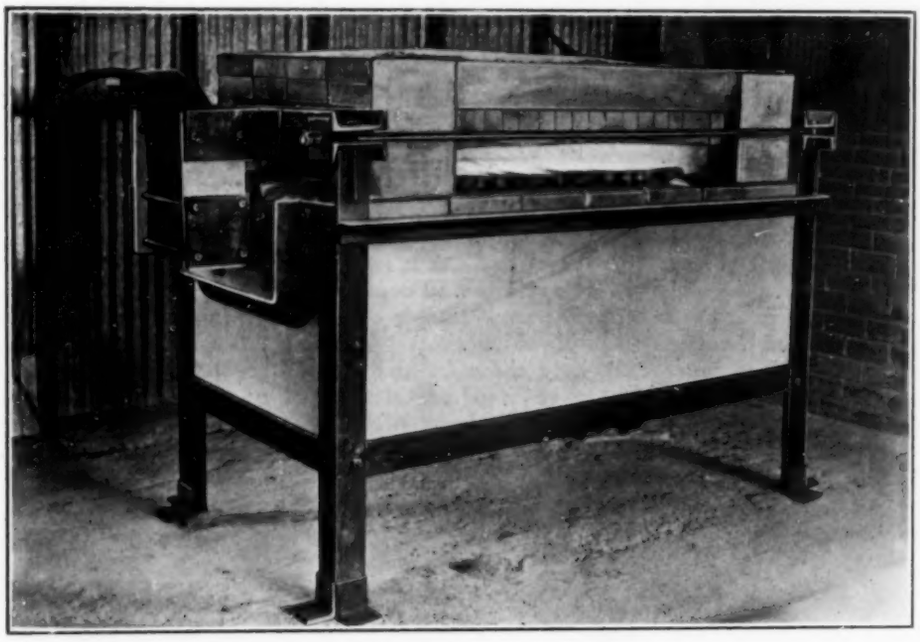

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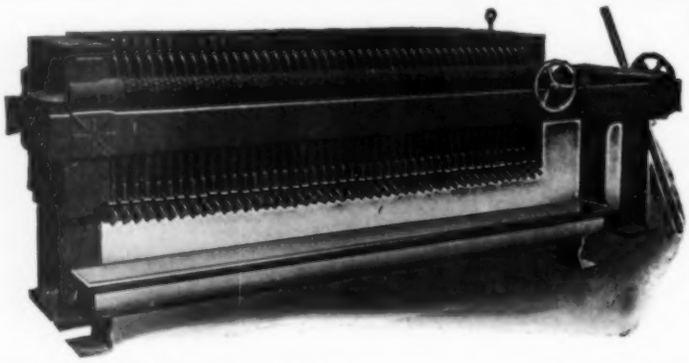
Index to Advertisers, page 71 Classified Directory, pages 68-69-70



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METALLURGICAL & CHEMICAL ENGINEERING

ISSUE OF JULY, 1913

Progress of Electric Smelting at Heroult, California. By John Crawford 383

The author, who is the plant manager of the Noble Electric Steel Company, at Heroult, Cal., gives here the first authentic and profusely illustrated account of the latest advances of this American pioneer plant for electric iron ore reduction. The relation of the electric furnace design and operation to the special needs of the Californian iron and steel market and the consequent differences from Scandinavian practice are pointed out.

The Relative Importance of Principles and Practice in Education. By Dr. James Douglas 377

His commencement address delivered before the Colorado School of Mines, with many exceedingly interesting reminiscences of early practice in the metallurgy of copper. The author emphasizes the pre-eminent value of principles in education and the necessity of fundamental investigations for future prosperity.

Manufacture of Petroleum Products. By Dr. F. C. Robinson 389

The author, who is chief chemist of the Atlantic Refining Company, gives in this paper, which is illustrated by diagrams, a concise sketch of the principal steps in the manufacture of petroleum products, with special reference to cracking distillation of crude oil and fractional distillation of crude oil.

Cyanide Practice in the Black Hills, South Dakota. By H. C. Parmelee 395

The first part of a sketch of present cyanide practice in the Black Hills of South Dakota, with references to proposed changes and improvements.

Purification of Blast Furnace Gases. By Camille Herwegh and Fred. H. Wagner 399

A report of a paper by Camille Herwegh giving data on the use of the Feld washer in the Pompey blast furnace plant in France, and an account by Fred. H. Wagner of a proposed American gas washer plant for two blast furnaces producing 5,400,000 cu. ft. of gas per hour, this gas washer plant being designed for continuous operation.

Metering and Recording the Flow of Fluids. By J. W. Ledoux 403

A paper on the principles of metering and recording the flow of fluids, with a description of new meters and recorders of the flow of liquids through Venturi tubes, orifices or conduits, based on the integration of the velocity head.

Evolution of Methods of Handling Slime—IV. By H. N. Spicer 408

The author, who is a high authority on slime treatment and cyanide practice, made last year an extended trip around the world, in which he studied the evolution of the methods of handling slime in different countries. The results of this investigation are given in this serial. In the present instalment the author deals with practice on the Rand, in South Africa.

EDITORIAL:
- Electric Iron Ore Reduction in California 367
- Testing Plants 367
- Steel Production Statistics 368
- O Tempora, O Mores 368

READERS' VIEWS AND COMMENTS:
- Electrochemical Spies in Niagara Falls. By Aug. Stillesen 370
- Cleaning Filter Cloths. By W. H. Weigand 370
- New York Hydroelectric Development and Niagara Falls. By Reginald Pelham Bolton, P. McN. Bennie, and F. Austin Lidbury 370-371

Western Metallurgical Field 371
The Non-Ferrous Metal Market 372
Iron and Steel Market 373
Application of Ozone to Water Purification 373
Water Power Decision of the U. S. Supreme Court 374
Electro-Analysis of the Copper Alloys. By J. G. Fairchild 380
The Modern Byproduct Coke Oven. By C. A. Meissner 382
British Iron and Steel Institute 388
Temperature Conversion Table. By Leonard Waldo 394
Notes on Chemistry and Metallurgy in Great Britain 412
Recent Chemical and Metallurgical Patents 414
Synopsis of Recent Chemical and Metallurgical Literature 416
The Adaptability of Electric Welding. By George Hills 419
Butte-Anaconda Electric Railway 420
New Regenerative Heating Furnace 421
Conversion of a Tube Mill into a Conical Mill 423
Personal 423
Digest of Electrochemical U. S. Patents 423
Book Reviews 424

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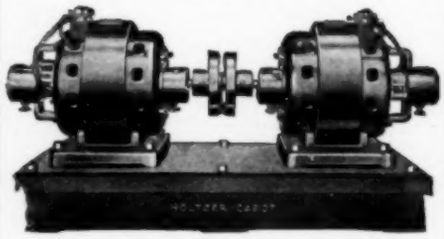

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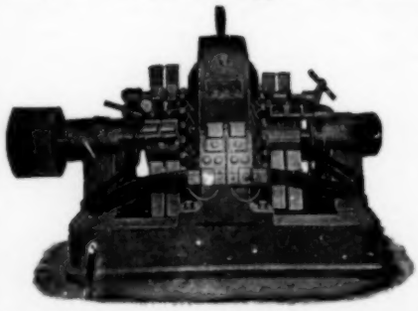

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CONTENTS.

Editorial:
Electric Iron Ore Reduction in California 367
Testing Plants 367
Steel Production Statistics 368
O Tempora, O Mores 368

Readers' Views and Comments:
Electrochemical Spies in Niagara Falls. By Aug. Stillesen 370
Cleaning Filter Cloths. By W. H. Weigand 370
New York Hydroelectric Development and Niagara Falls. By Reginald Pelham Bolton, P. McN. Bennie, and F. Austin Lidbury. 370-371
Western Metallurgical Field 371
The Non-Ferrous Metal Market 372
Iron and Steel Market 373
Application of Ozone to Water-Purification 373
Water Power Decision of the U. S. Supreme Court 374
The Relative Importance of Principles and Practice in Education. By James Douglas 377
Electro-Analysis of the Copper Alloys. By J. G. Fairchild 380
The Modern By-product Coke Oven. By C. A. Meissner 382
Progress of Electric Smelting at Heroult, Cal. By John Crawford 383
British Iron and Steel Institute 388
The Manufacture of Petroleum Products. By F. C. Robinson 389
Temperature Conversion Table. By Leonard Waldo 394
Cyanide Practice in the Black Hills, South Dakota. By H. C. Parmelee 395
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Metering and Recording the Flow of Fluids. By J. W. Ledoux 403
Evolution of Methods of Handling Slime.—IV. By H. N. Spicer 408
Notes on Chemistry and Metallurgy in Great Britain 412
Recent Chemical and Metallurgical Patents 414
Synopsis of Recent Chemical and Metallurgical Literature 416
The Adaptability of Electric Welding. By George Hills 419
Butte-Anaconda Electric Railway 420
New Regenerative Heating Furnace 421
Conversion of a Tube Mill into a Conical Mill 423
Personal 423
Digest of Electrochemical U. S. Patents 423
Book Reviews 424

Electric Iron Ore Reduction in California

It is with great pleasure that we publish in this issue the first authentic report of the latest advances in the pioneer work of the Noble Electric Steel Company in Shasta County, California, in electric iron smelting. California and Scandinavia were the pioneers. Both started almost simultaneously, but independently, and developed independently, and as Mr. Crawford's most interesting article shows, the differences between Californian and Scandinavian practice and design have increased lately more and more, so as to suit local conditions. Now that electric iron smelting has become an established industry of this country the congratulations and good wishes of all interested in electro-metallurgical progress are due to the sturdy Californian pioneers and to their leader, Mr. H. H. Noble.

Testing Plants

The action of Colorado's governor in vetoing an appropriation of $40,000 for the completion and operation of the ore-dressing and metallurgical laboratory of the Colorado School of Mines calls attention to the proposed functions of that department of the school, and to the limitations which the governor's action will place upon it. In his own defense, the governor claims a lack of funds to meet the appropriation; but the governor, be it known, is distinctly an agriculturist, and as such possibly lacks an appreciation of the needs of a mining school, especially in a state where agriculture has temporarily overshadowed mining.

At present the experimental laboratory at the school is only partly equipped. It was designed for a two-fold purpose—education and investigation, but not for ore testing in the ordinary sense of that word, meaning the treatment of an ore by known routine methods and merely reporting the results, as is done by many commercial ore-testing plants. In its educational function it is intended to afford a laboratory for the students working under the direction of their instructors. This is practically all that can be done during the next biennium, in view of the lack of funds for completing the plant and providing for metallurgical investigations. Instruction can be given in ore dressing and cyaniding, but until an adequate heating equipment is installed even this work must be curtailed during the winter. The net effect of the lack of funds will be to defer the efficient use of an admirable laboratory for two years.

Perhaps the larger function of the plant is to afford independent engineers a place to investigate methods and processes and test new inventions. Ordinary facilities for power and ore-handling machinery are at hand and there is ample space for the temporary construction and use of any device that might be needed. In this respect also the success of the plant must be deferred.

Elsewhere in this issue we note the entrance of the Canadian government into the ore-testing business. A com-

plete ore-dressing and metallurgical laboratory has been opened at Ottawa, where Canadian ores are to be tested free of charge. This laboratory also is to supply Canadian engineers with facilities for investigating problems in ore treatment, and arrangements can be made for the use of the laboratory by those who wish to undertake any research. The establishment of this laboratory is a departure in governmental affairs. Our own Bureau of Mines has taken no such step and probably will not. For a government to enter "business" is a dangerous step.

Steel Production Statistics

Official statistics of the production of steel ingots and castings in the United States have been published for 1912 by the Bureau of Statistics of the American Iron and Steel Institute. The output, in gross tons, has been as follows:

	1910.	1911.	1912.
Ingots,	25,154,087	23,029,479	30,284,682
Castings	940,832	646,627	966,621
Total	26,094,919	23,676,106	31,251,303

As 1911 was distinctly an "off year" in the iron and steel industry it may merely be remarked that there was an increase in steel output from 1911 to 1912 of 32 per cent. Of more interest is the comparison of 1912 with 1910, which was easily a record year, passing any previous year by 8 per cent. The gain from 1910 to 1912, however, was 19.8 per cent. Passing to more long-range comparisons, we observe that 1906 was a year of full output throughout, the steel mills being taxed to make deliveries, yet 1906 showed an output of only 23,398,136 tons, the gain of 1912 over that year being more than 33 per cent. Yet it is to be borne in mind that 1912 was not a year which seemed to bring out the full consumptive requirements of the country, nor did it bring out the full productive capacity of the country, though it came close to doing so. The idea that the steel industry has not grown much in the past half dozen years must be dismissed.

It is interesting to note that the production of steel ingots and casting in 1912 exceeded the production of pig iron. This was the case by a small margin in 1911, but in 1912 the margin was large, no less than 1,524,366 tons. There was, of course, a large tonnage of foundry grades of pig iron produced. The actual output of Bessemer and basic pig iron and of spiegeleisen and ferromanganese in 1912 was 23,303,625 tons, and neglecting the tonnage of manganese bearing metals imported, there was an excess of 7,950,000 tons, or 34 per cent. in the steel ingots and castings produced over the blast furnace material consumed. This brings us to the point that the "production of steel ingots" as commonly reported is that, and nothing more. It is the production of ingots, not of commercial steel. It is scrap which makes up the difference, and more too, since there is an absolute loss of iron and of metalloids in the conversion into steel, but this scrap is largely self-produced, the material being produced in rolling, and eventually passing a second time over the ingot scales. There is, of course, a tonnage of old material used, but this is easily the smaller factor.

Recent gains in steel production have been confined entirely to the basic open-hearth process. Bessemer steel has not attained its record made as far back as 1906, acid open-hearth has been practically stationary for a decade and crucible steel has recorded no clear gain, while the electric steel tonnage (18,309 tons in 1912) is too small as yet to be a commercial factor. The Bessemer output in 1912, 10,327,901 tons, fell short only of the output in 1905, 1906, and 1907, and the loss from the record year, 1906, to 1912, was only 15 per cent. Considering that meanwhile Bessemer steel has practically disappeared as material for structural shapes, has disappeared as plate material in the case of nearly all important work, and last year contributed only one-third to the total rail tonnage, it is evident that it is still much in requisition for other purposes, else it could not stand these losses without showing up quite poorly as to total tonnage.

"O Tempora O Mores"

The most modern philosopher, Henri Bergson, talks about intuitive consciousness and about class-consciousness. This philosophy so much in vogue can be turned in criticism against those who like it and the terms class and age "self-consciousness" applied. This age in its attitude of self-consciousness calls itself "the age of unrest," as if this was the only age in which the waters were always and everywhere in motion and as if society were heretofore in a static position and as if social kinetics were not actually the normal state.

But we would fain accept this description of title of "the age of unrest," for it shows so clearly the mental pose, not the mental poise of the troubled. Painting and sculpture have of late become markedly pornographic. Letters have assumed the picaresque. Social life has been colored with an unnative hue of extravagance of thought and action. Its paganistic tendencies are seen in the immense consumption of little white cylinders which emanate in a state of slow combustion, the meretricious incense of the Orient. The youthful female portion correspondingly seems to like to dress like Egyptian dancing girls. Puritanism has so waned that turkey-trotting is almost tolerated in Boston.

The condition exudes one-third sensualism, two-thirds egotism. Certainly, there are countercurrents strong and deep of balance, moderation, good taste, and intelligence. But the troubles bubble from the cauldron showing the start of some great refining action like in an open-hearth furnace when the pig and ore begin to "boil." Now the process from crude pig iron to refined steel is a turbulent one.

These troubles must be largely mental in their character, for people at no time were so well fed, so well clothed and so well protected against the elements. The inherent value of troubles to the world lies in the mental stimulus that they impart. But mental troubles are due in large part to the false and insecure base-line from which sociological conceptions are measured.

It will be found interesting to trace these phenomena on the analogy of quantitative molecular physics to their immediate causes. When a new force comes into any chemical reaction and plays no rôle except to accelerate the reaction-velocity, it is called a "catalyser." A catalyser can be sunlight, platinum black, iron oxide, an organic ferment or an alternating current of electricity. Things are moving

so fast that some catalyser must be working on society. Let us ask science what is this sociological catalyser. The answer comes from science itself, for the catalyser is science.

Science has reacted in two ways on human life. First, it has thrown up a new superstructure of facts of nature and so has profoundly affected theological thought or, better expressed, theological talk. Second, in its applied form it has created material wealth rapidly and made money-making very easy. In ordinary New Yorkese, "a big fat pie has been thrown before us and the angel-faced ones are scrambling for the best of it." To be concrete, the telephone, the telegraph, the motor, the modern high-speed tool steel, all the numerous products of the chemical and physical laboratory have made the efficient man vastly more efficient industrially speaking. He gets more, spends more, and the lesser ones envy more. Just this describes it, "getting and spending we waste our powers."

It has recurred to our mind that the advent of the flying machine is a typical case explaining well one phase of the phenomena. Nothing is so marked to-day as the disrespect in which the young hold the old. For a long time the older ones had been telling the younger ones that flying was the "dream of dreams," and even the wise elder said that it was impossible to make an aeroplane, since man weighed 150 lb. and the condor, largest of winged fowls, weighed only 70 lb. In parenthetical thought, let us acknowledge that their reasoning was perfect, but that the advent of the 1-hp-per-3-lb. gasoline motor was an unforeseeable fact.

Now, if you tell a child many times that things cannot ever happen and are wrong everytime, the respect of the child for you will surely approach zero as its limit. This has happened in the case of aeroplane development universally and has been the case in many, many other instances. It is hardly to be denied that we should expect disrespect from the young if we say that something never can be done and the children when they see that we are wrong get an enlarged, imaginatively stimulated egotism. "Most kids nowadays get the big head."

In the rich man's panic of 1903, the late Pierpont Morgan aptly described things as suffering from "undigested securities." His words can be applied in paraphrased form that there are undigested mental securities present in our life.

But, the call of the fates breaks loudly on our ears, "where are we at?" and "whither are we going?" We hear the plaintive minor key and then the strong major chord. History teaches that the world usually gets somehow what it really wants and needs. So much is clear. What the world needs now is men, not monkeys—men who can see, think, and act. History likewise teaches that the dumb, inarticulate cry of the many is heard by the few, who interpret it as a mother does the cry of her child, and by intelligence removes the cause. May we not divine that to the groans and rebellious sobs of the weltering masses will come the help of the true aristocrat, he who rules with love?

These mental securities of the present day seem paradoxical, they are so insecure. The latest science coming to a quantitative basis is psychology, which any scientist will call the greatest of sciences. When a science gets on an exact basis, it can be quickly applied to practical uses, for "there is safety in numbers." Signs are not now wanting that there will be an application of psychology to man's needs. Of course, applied psychology will be first mixed with a large proportion of charlatanry, just as was the case with alchemy. Probably there will be an intensification of practical psychology in religion. Both these will be of value. A good many sensible people are of the opinion that the present day so-called Christianity contains too much that is sloppy and sentimental and that its strong leaven has been weakened in nineteen centuries. Would it also not be better if the preachers should "render unto Cæsar the things that are Cæsar's and to God the things that are God's," for indeed a strict law of psychology is that values of one kind can only be measured in a unit of that kind?

As Charles Proteus Steinmetz says, mathematics can be applied by analogical methods to all branches of human knowledge. In our daily mental life, the curve $y = \sin x$ or a mathematical expression of periodic phenomena applies as an analytical means to the solution of problems as surely as it does to music. The law is that things that start slowly to change, increase, reach a "temporary ultimate," slowly wane, then go on the descending curve. To apply usefully this equation of the law of compensation to the needs of man is one of the great problems of practical psychology.

The interpreters of modern thought must be master minds, for the world of men, women, and things is governed by intellectual Czars as much in these days of boisterous democracy as in olden times. In the intellectual world there is perfect freedom, tho' or since, as one views the question, its laws are inexorable. Bergson now dominates world philosophy, succeeding Spencer. So in the general catalysis produced by science, we see the equation $y = \sin x$, where the crest of the wave of materialism was reached some fifteen years ago and the imaginative and emotional spiritualism of science has started. Indeed the vogue that Christian Science has received is largely due to an instinctive popular revolt against deriving all action from materialistic causes, quite sufficient and proper in their own sphere but not of wide range. In the change among purely intellectual people, the rationalistic common-sense philosophy of William James, called "pragmatism," has been prepotent.

It seems reasonable to believe that things will change, not in one way but in another. At all events, one thing is sure, we have life and nature before us, with us, and of us. No one can take a survey of the country from a mountain top, forget himself, think of the universe and help but say, "in the twentieth century we do not exist, *we live*." If he forgets his weak, pitiful self, he will be of use to the world and become active and truly happy. As a great technical chemist says, "happiness is purely subjective."

The general catalytic effect of science on human thought and life may be likened to the incantations of the witches of Macbeth, "Toil and Trouble boil and bubble, Cauldron, boil and bubble." It is a firm deduction that science is the cause and that science in its widest sense will be the result. All the above is as plain as the nose on the face of a plain girl.

Readers' Views and Comments

"Electrochemical Spies in Niagara Falls"

To the Editor of Metallurgical and Chemical Engineering:

Sir:—On page 235 of the May issue of your journal is printed a note entitled "Electrochemical Spies in Niagara Falls." As this note is incorrect, allow the following facts to be known.

Mr. Kvernoe was not in the employ of any foreign concern at his stay here in this country. The statement that money was plentiful and freely used is also incorrect, as Mr. Kvernoe was dependent for his living on what he could earn by his work here. The statement that two accomplices of Mr. Kvernoe went about the country is a product of your informant's brisk imagination, to say no worse, and that a Swede and German were his accomplices is nothing but pure fabrication on the part of your informant.

What Mr. Kvernoe can be reproached for is that, when asking for work, he did not mention he had technical education. Mr. Kvernoe has now gone back to Norway which fact, it is to be hoped, will cause the return of your informant's equanimity.

Aug. Stillesen.
Norwegian Vice Consul.

Niagara Falls, N. Y.

Cleaning Filter Cloths

To the Editor of Metallurgical and Chemical Engineering:

Sir: I observe in the May issue of Metallurgical and Chemical Engineering editorial mention of methods of cleaning filter cloths, in which the use of steam under low pressure is said to have been introduced into the Black Hills by Mr. F. C. Bowman. This is in error, since I had made use of the same agent for cleaning the leaves of a clarifying filter of lime deposit during the winter early in 1912, at the same mill of which Mr. Bowman now has charge.

The idea did not originate with me, as I had seen steam used to dislodge lime accretion from the interior of solution pipes some years previous to my use of it on the clarifying leaves mentioned.

Regarding the use of steam on filters of the drum type, my experience has proved to me the futility of using it on the discharge port only; the interval of time in which the steam has to act is so short, before the leaves pass into the pulp again, that condensation takes place before the steam has time to act with much effect. A much better scheme would be to have the piping so arranged that—the vacuum being entirely off—the steam could be introduced into all the leaves simultaneously, thus obviating the counter effect of condensation. The time of treatment would also be much shortened.

There is no reason why steam could not be used on the ordinary form of leaf filter, especially where arranged in small sets or baskets. The steam need be used only a few minutes after the leaf is permeated, when water, slightly warmed, may be introduced and the softened lime is readily washed out.

The use of warm mill solutions would do much to prevent excessive deposits of lime in cold weather. W. H. Weigand.

Trojan, South Dakota.

New York Hydro-Electric Development and Niagara Falls

To the Editor of Metallurgical and Chemical Engineering:

Sir: Your issue of June contains two communications referring to my recently published book, "An Expensive Experiment," the points in which are of such general interest that I hope I may be permitted space for a reply.

Taking first the letter of Mr. Bennie, I would point out to him that the dilemma in which he has imaginatively placed me will be found upon examination to have no existence. The old photograph, taken before the power development, is evidently one showing the extreme low water condition of 1895, and fully confirms the point made by the use of the official recent photograph in my book, which shows that the same results are brought about now by the diversion of water by the power companies at a time of not unusually low water flow. I regard the old picture, therefore, as confirmatory of my statement of the situation, and if I had been able to secure such a picture at the time of publcation I would have used it for this purpose.

The criticism of Mr. F. A. Lidbury appears to be influenced by his appreciation of the value of the electrochemical industries which have been brought into being around Niagara Falls. I share with him, and have expressed in my book this appreciation of the utilization of hydro-electric energy in such industrial processes, but I do not believe that he or any other student of the subject will dispute my point that the partial utilization of hydro-electric energy in general electrical distribution is not economic. If the water of Niagara Falls is to be in whole or in part diverted from scenic utilization then it should be for the purpose of adding to the financial wealth of the community, and this is not to be done by dissipation in partial and intermittent use for light and general power purposes.

The reply made by Mr. Lidbury to the report of the Committee of Engineers of the United States Government, as to the indirect value of scenic Niagara, is not to the point. It is only necessary for us to imagine the result of the entire diversion of Niagara from its present course into power plants to realize the ruin that will fall upon a large portion of the surrounding communities and the losses which would spread far and wide therefrom.

I cannot, therefore, agree with your reviewer that this indirect source of business is a fallacy. It is satisfactory, however, to find that all of us are in agreement upon the main issue—the undesirability of the expenditure of State funds in hydro-electric developments, the commercial value of which is evidently a subject of doubt in a number of cases, and which, by the progressive improvements in other directions of power production, may eventually lose much of their practical value.

The State of New York is to be congratulated upon the firm attitude adopted by its Executive in disposing of the recent attempt to fasten upon it an ill-considered policy of this chcaracter. Reginald Pelham Bolton.

New York City.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Referring to Mr. Bolton's communication anent the photograph shown in your June issue, I do not in anywise agree with his notion that his dilemma vanishes upon examination. His difficulty is still as substantial as the rocks behind the falls themselves, notwithstanding the spray within which he seeks to conceal it.

It is not well to permit the actual point at issue to become befogged. Mr. Bolton's book contains a photographic reproduction of a portion of the falls, alleging "An Effect, in Part, of the Expensive Experiment." This I deny, and in support of my contention show another photograph "at least 20 years old." Mr. Bolton immediately jumps to the convenient and (for him) useful conclusion that my photograph is "evidently one showing the extreme low water condition of 1895" and proceeds to reach some "therefores," which unfortunately are still at variance with the facts, and "therefore" unjustified.

My photograph is not that of the extreme low water condition of 1895. It must be obvious to your readers that 1913, minus "at least 20" does not result in 1895. Of the two gentlemen concerned in the taking of the original photograph, but one sur-

vives, and he assures me that it is even older than 20 years; how much older, I need not disclose to prove my point.

Mr. Bolton's claim that he would have used such a photograph had he been able to obtain a copy seems rather disingenuous in view of the fact that it is really but a portion of one of the standard photographs of Niagara Falls, taken as typical of its beauty. Most visitors to Niagara Falls find it difficult to avoid carrying a copy home with them, due to the importunate blandishments of numerous souvenir bazaar keepers. I assure him he has my permission to obtain all the copies he wants, but he should be careful how he uses them.

With the fog thus dissipated by this discharge of fact, his dilemma clearly reappears.

Niagara Falls, N. Y. P. McN. BENNIE.

* * *

To the Editor of Metallurgical and Chemical Engineering—

Sir:—Mr. Bolton's reply to my criticism of his letter to the *Electrical World* requires but little comment.

It is as easy to attempt to dispose of an argument by stating that it "is not to the point" as it is to attempt to avoid a dilemma by calling it an imaginary one—and about as convincing.

Unaccountably enough, "the surrounding communities" do not seem to "realize the ruin" which has fallen upon them as a result of the present diversion of the Niagara river, nor is there any evidence that a considerable extension of the same kind of "ruin" and "losses" would be otherwise than welcome!

It would be difficult to gather from Mr. Bolton's book, and impossible to gather from Mr. Bolton's letter to the *Electrical World*, that "appreciation" of the economic value of hydroelectric power as applied to the electrochemical industries to which he now gives somewhat belated expression. It is to be hoped that, when next the question of utilization of Niagara Falls engages his pen, he will endeavor to show that he has at least a general acquaintance with the facts, and a desire, in drawing conclusions, to adopt the dispassionate and unprejudiced attitude which is expected of an engineer.

Niagara Falls, N. Y. F. AUSTIN LIDBURY.

The Western Metallurgical Field

Effect of Spelter Prices on Production

The drop in the spelter market from approximately 7 to 5 cents per pound for metal has had a decided effect on production in the Western zinc-producing districts. Ignoring for the moment the causes contributing to the low market price, the fact remains that many of the smaller producers and some of the larger ones find themselves either unable or unwilling to mine and concentrate zinc ore under prevailing conditions. In Colorado zinc production has practically ceased at Leadville, except for some production of carbonate ore to fulfill contracts; at Rico the output has fallen off decidedly, and at Creede the production is almost without profit. The Wellington, at Breckenridge, has curtailed production to about one-third of the normal.

The condition in other states is not unlike that in Colorado. The Yellow Pine in Nevada has ceased zinc production, as has the Federal at Wallace, Idaho. Butte & Superior, at Butte, Mont. one of the largest western zinc producers, is contemplating closing its mill until market conditions improve. At Joplin, Mo., the American Zinc, Lead and Smelting Co. has closed three of its large mills; and other companies operating in sheet-ground, where the ore is low grade and difficult of extraction, have discontinued operations.

Popular opinion generally ascribes the spelter situation to prospective tariff legislation. It is doubtful, however, if this assumption is correct, for when we consider the enormous increase in production of spelter during the past two years, and the rate at which it was bought by consumers, the impression will gain ground that buyers became oversupplied and ultimately ceased buying, with the inevitable result of depressing the price for spelter. The market is now below the average, and this has had the effect of curtailing production which will continue until more nearly normal conditions obtain.

Oregon-California Mining Congress

The recent session of the Southern Oregon and Northern California Mining Congress, at Redding, Cal., was one of the most interesting and profitable meetings of that organization. Considerable attention was devoted to the subject of smelter fumes, and the effect of recent litigation on the smelting industry of California. One of the first acts taken was to appoint a committee to visit the agricultural districts surrounding the smelters, and make a personal inspection of conditions in the field and observe alleged damage from smelter fume. The report of the committee was to the effect that a very careful and exhaustive examination showed all agricultural crops in good condition; that while smelter fume had in the past been responsible for some damage, nevertheless it seemed improbable that any such damage was now being done by any fumes escaping from the Mammoth plant at Kennet. The report further deprecated the conflict waged against smelting by the farmers, and expressed the opinion that the present inimical attitude of the latter was unwarranted.

The warfare of California farmers against the smelters reached its height some time ago when the agriculturists succeeded in getting a grand jury to declare the Mammoth smelter a public nuisance. By this means they hoped to unload the burden of further expensive prosecution, and place it on the authorities of Shasta county. The latter, however, have wisely concluded not to take any steps unless the farmers supply them with adequate evidence on which to base a suit, and the conspiracy is likely to fail.

It has already been suggested in this journal that the best solution of the smelter fume question is to refer complaints to a board of arbitration or some commission which shall control the dispute between farmers and smelters, determine the conditions under which smelters may operate, and make awards for damage if any is sustained. Such an arrangement already exists between the United States government and the Anaconda Copper Mining Co., and a similar scheme is under way for the settlement of the controversy between the Selby smelter in California and the inhabitants of Solano county. There is no reasonable excuse why such an equitable arrangement should not be concluded between the farmers of northern California and the smelters in that part of the state, for the present attitude of the agriculturists is intolerable and seriously interferes with legitimate smelting operations.

Canadian Government Testing Plant

According to a bulletin recently issued by the Department of Mines, Canada, the government has established a well-equipped ore dressing and metallurgical laboratory at Ottawa, under the direction of Dr. Eugene Haanel. The laboratory is established for the purpose of experimenting on the concentration and metallurgical treatment of Canadian ores, and is equipped with apparatus in both commercial and laboratory sizes.

The standard size machinery offers facilities for crushing and screening, sampling and recording, amalgamation and concentration. The crushing machines are one each, 12-in. by 8-in. Blake crusher, 24-in. by 14-in. rolls, and 4-ft. 6-in. Hardinge conical ball mill. For screening a 6-ft. Ferraris screen is used for coarse sizing, a Keedy sizer for fine sizing and a duplex Callow screen for concentration work. Sampling is provided for by two standard Vezin machines placed in favorable position to cut out preliminary samples of coarse material. An 8-unit system of Flood automatic samplers is provided for fine material. Hand sampling can be accurately done by means of Jones riffles. All water lines are equipped with Keystone water meters to keep accurate records of water consumption.

For amalgamation and concentration the equipment includes a 5-stamp battery with 1250-lb. stamps, a 10-ft. amalgamating table and Pierce amalgamator. The mortar of the stamp mill may be arranged for inside amalgamation. Six 8-ft. Callow tanks are available for desliming and settling; two Richards pulsator classifiers of the launder type, one Richards pulsator two-compartment jig and one Overstrom sand table and Deister

slime table for concentration. Magnetic and electrostatic separation can be accomplished on several types of machines: a tandem unit Grondal magnetic separator for wet separation of strongly magnetic minerals; a Grondal magnetic cobber for dry separation of strongly magnetic minerals; an Ullrich four-pole magnetic separator for either wet or dry separation of weakly magnetic minerals; and a Huff electrostatic unit, consisting of a standard generator and two laboratory-size separators.

The laboratory units consist of Sturtevant crusher, rolls and screen; Braun pulverizer; Abbe 6-jar pebble mill; gyratory screen for grading analyses; Richards combined laboratory pulsator jig and classifier, with glass side; Grondal separator for either wet or dry work; Wilfley table; cyanide plant of 200-lb. capacity, consisting of a Parral agitator and air pump, with necessary tanks; two filter presses, and complete sets of I. M. M. and Tyler standard screens for grading purposes.

It is expected that the installation of a roasting and sintering plant will be undertaken this year. The entire plant will be operated free of charge on Canadian ores, under the following conditions: (1) Samples must be sacked and delivered to the plant free of all transportation charges. (2) Not less than 200 lb. will be accepted for small-scale tests, nor less than 5 tons for large-scale tests. (3) All testing products become the property of the Department of Mines, unless stipulation to the contrary is made before the test begins. (4) Reports of tests will be incorporated in the publications of the Mines Branch, but single copies will be given to the owner of the ore tested. Ordinarily the tests will be made by officials of the department, but arrangements can be made for competent engineers to supervise their own tests. It is expected that the plant will be ready for operation early in the month of July, 1913. All communications regarding its use should be addressed to Dr. Eugene Haanel, Director Mines Branch, Department of Mines, Ottawa, Canada.

Concentration at Cripple Creek

It is well known that the fine material resulting from crushing many of the ores of the Cripple Creek district is comparatively richer than the average ore. This is due to the fact that the planes of fracture in crushing are coincident with fine veins of high-grade mineral which become more or less pulverized in the operation. Various schemes have been adopted in the Cripple Creek district to save this portion of fine, rich mineral. Dry screening, or washing, or combination of the two, have been used. The Golden Cycle company uses dry screening, and the Elkton company has a wet washing plant. Recently the Vindicator company has decided to erect a concentrating plant, and tests are now in progress to determine the best method of procedure. It is probable that both dry and wet processes will be used in this plant, and the product will be shipped to smelters. Considerable of the fine mineral can be recovered by dry screening, and this may be followed by some form of wet dressing to recover the portion that requires attrition to be removed from the rock faces.

Company Reports

The Anaconda Copper Mining Co. presents some interesting facts regarding its operations in the annual report for the year 1912. The year was one of the best in the history of the company, and resulted in the mining and treatment of a larger tonnage of ore than for several years previous. With the marked improvement in the price for copper, the company entered into an agreement with the Butte Miners' Union to provide for an automatic variation in the wage scale to correspond with the fluctuations in the price for copper. The terms of the agreement provide: that when the monthly price for electrolytic copper is 15 cents and over, but under 17 cents, the wages of all men employed underground shall be increased 25 cents above the minimum wage of $3.50 per day; and that when the price of copper is 17 cents and over, the wages for underground men shall be increased an additional 25 cents per day. The agreement is such that the contract is automatic in its operation, and that corresponding reductions in the increased wage will take place when the price of copper falls below the figures named. Similar contracts were entered into with different classes of labor other than the above. All contracts expire on June 1, 1915.

The reduction works treated during the year 3,880,203 tons of ore and cupriferous material at Anaconda, and 1,189,039 tons at Great Falls. Of this, 4,486,873 tons came from the company's mines, 581,032 tons were purchased, and 1338 tons comprised precipitates and cleanings from the Old Reduction Works. The production of metals was as follows:

	Lb. Copper.	Oz. Gold.	Oz. Silver.
Anaconda	222,763,670	52,564	9,702,605
Great Falls	71,710,491	61,314	11,014,737

The report chronicles the attempt to develop a successful centrifugal concentrator for slime, experiments with which began in 1910. Several machines have been built in an effort to overcome mechanical difficulties, and one machine ran throughout the year 1912. While the results indicate that for certain classes of material the operation of the machine is satisfactory, there is still a doubt whether for everyday practice some one of the other machines tested might not be of equal service.

One of the sections of the Washoe concentrator was remodeled to conform to certain details of practice which had been evolved at Great Falls, and the results based on a test for three months proved it to be an absolute success.

Various experimental processes have been tested during the year, with varying success. These processes had in view improvements in concentration and the reduction of slimes and tailings.

The process for slimes promises well and will be tested on a large scale. Should it prove successful, a complete installation will be made, which will mean not only a reduction in costs, but a marked increase in output of refined copper from the same tonnage of original ore treated. The process has been perfected by employees of the company, and will be useful in treating the accumulation of many millions of tons of tailings which are available at the Washoe plant.

The Great Falls smelter will be rebuilt during 1913. The old blast furnace department, flues and stack, together with electrolytic refinery, will be retained, but the balance of the plant will be rebuilt. The greatest improvement made during the year was in the converting department, where a radical change was made in the size of converters. The "Great Falls" type of cylindrical converter has been built as large as 20 ft. in diameter, and the new smelter will be equipped with converters of this type.

The Non-Ferrous Metal Market

Prices for the non-ferrous metals during June have been quite unsettled, with a tendency to a steadily lowering market. But little business has been done, and the lack of large prospective orders probably has kept prices from going still lower on competitive bidding. Unsettled financial conditions have been reflected in metal prices, and consumers have been unwilling to buy heavily under the conditions.

Copper.—This market has been showing little life, and the liquidation of speculative holdings has tended to lower prices. The latest quotations are 14¾ to 15 cents for Lake, and 14.70 to 14.80 cents for electrolytic.

Tin.—Prices have declined steadily since our last report, and the market has been weak. Liquidation has continued, and has resulted in the quotation of spot tin below three months—something that has not occurred for some months. The last quotation is about 45¼ cents for June tin, New York.

Lead.—The market has fallen off somewhat and for the past two weeks has been unchanged at 4.17½ to 4.20 cents, St. Louis, and 4.30 to 4.35 cents, New York.

Spelter.—Despite the lower prices business in this metal has been smaller in volume than for some time. Production from some Joplin and Western mines has ceased, as the market affords no profit. The last quotations are 4.85 to 4.95 cents, St. Louis, and 5 to 5.10 cents, New York.

Other Metals.—The market for aluminum has been quiet, and little business has been transacted. Prices are 25½ to 26 cents for spot and 26 to 26½ cents per lb. for future delivery. Prices for antimony vary from 7.50 to 9 cents per lb. for various brands. Business in quicksilver is fair and prices are steady at $40 per flask of 75 lb., New York, and $39.50 at San Francisco.

Iron and Steel Market

June has been a somewhat quieter month than May in the finished steel trade, but production has been continued at substantially the maximum rate, the mills running chiefly upon old orders. The large amount of business which had been entered upon books has stood the test of a dull market very well, there being an almost negligible amount of cancellations and postponements.

Strictly new buying in June probably did not equal 50 per cent of the shipments, but the rate of buying was no smaller than in May. Specifications against old contracts averaged 60 or 65 per cent in June, against 70 or 75 per cent in May, showing a decrease which was to be expected as midsummer approached.

The present position is that the large mills are booked up almost fully, with perfectly trustworthy business, until nearly the close of the year, in rails, plates, structural shapes and merchant steel bars. In tin plates they have actual specifications for from 6 to 12 weeks, and in sheets and tubular goods for from 4 to 10 weeks. In wire products there is practically no business actually specified on books and specifications are coming in slowly on contracts.

The smaller mills, comprising perhaps 20 per cent of the total capacity, have relatively little business on books. Normally they do not book up extensively, endeavoring rather to obtain premiums for prompt shipment, and in this they succeeded well in the early months of the year. Now the premium business has disappeared and it is even reported that some mills in the east are shading structural shapes and plates by $1 a ton.

With so large an amount of business already on books, it would require only a relatively small buying movement to fill the mills completely for the balance of the year. Such a buying movement is predicted and by some as early as next August. It is represented that by that time good crops will be assured and the tariff bill will probably be out of the way. So far as regards this year's business, the iron and steel industry objects to the tariff revision chiefly on account of the sentimental disturbance given to general business, though for the general future it is held that some of the readjustments required by the prospective tariff will cause trouble.

Steel prices have remained firm on the whole. Premiums for prompt delivery, of course, disappeared some time ago, but the regular mill prices have declined in only two instances from the top point reached in last year's rise, these exceptions being sheets and wire products. The latter are easily shaded $1 or $2 a ton, and a general revision in wire prices is likely to occur shortly, such revisions being common about July 1. Several months ago galvanized sheets declined $2 a ton, and later black sheets declined $1, while another decline of $1 has occurred in the past month in black sheets. Manufactured goods, such as shafting, rivets, spikes, etc., have experienced considerable declines in the past few months.

Pig Iron

Early in June distinct hopes were expressed that the decline in pig iron was over and that more active buying would ensue, with a consequent rise in prices. These hopes have been dispelled. Pig iron has declined throughout the country an average of about 50 cents during June, against a decline of about $2.25 during the preceding five months. There has been slightly heavier buying, but instead of this buying supporting the market it has tended if anything to weaken the situation, through buyers taking relatively small tonnages and then withdrawing, so that the market loses the prospect of their buying in a liberal way in the near future. The decline throughout has been one actuated largely by sentiment, buyers refusing to make forward purchases as long as they observed a declining tendency, and at no time has the market had sufficient vitality to invite a general buying movement. Stocks have accumulated in a small way in some districts, but in general they are light. Production of merchant iron has been curtailed slightly in the past two or three months, but a greater curtailment is in prospect during the next two months. The market is now quotable as follows: Southern No. 2 foundry at Birmingham, $10.75; No. 2 X foundry, Philadelphia, $16; Buffalo, $14; No. 2 foundry, Cleveland, $14.75; Chicago (at furnace), $15.50; at Valley furnaces (90 cents higher delivered Pittsburgh), Bessemer, $16; basic, $14.50; foundry, $14; forge, $13.75; malleable, $14.25.

Steel

The scarcity of unfinished steel has continued and efforts by consumers to break the market have been largely unavailing. There is no demand to test the market for prompt sheet bars, but prompt billets bring premiums of $1 to $2 a ton or more, depending on size and analysis. The regular forward market is quotable at $26.50 to $27 for billets and $27 to $27.50 for sheet bars, f.o.b. maker's mill, Pittsburgh or Youngstown. Rods have declined $1 a ton to $29, Pittsburgh.

Finished Steel

Regular prices are as follows, f.o.b. Pittsburgh, unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.45 cents.

Shapes, 1.45 cents.

Steel bars, 1.40 cents, base.

Common iron bars, 1.65 cents, Pittsburgh; 1.57½ cents, Philadelphia; 1.50 cents, Chicago.

Wire nails, $1.75 per keg, base, nominally, $1.80; plain wire, 1.60 cents.

Sheets, blue annealed, 10 gage, 1.75 cents; black, 28 gage, 2.25 to 2.35 cents; galvanized, 28 gage, 3.40 to 3.50 cents; painted corrugated, 28 gage, 2.45 to 2.55 cents; galvanized corrugated, 28 gage, 3.45 to 3.55 cents.

Tin plates, $3.60 for 100-lb. cokes.

Merchant steel pipe, 79 per cent off list for ¾ to 3 in.

Steel boiler tubes, 3½ to 4½ in., 69 per cent off list.

Standard railroad spikes, 1.80 cents, Pittsburgh or Chicago.

Button head structural rivets, 2 to 2.10 cents; cone head boiler rivets, 2.10 to 2.20 cents.

Application of Ozone to Water Purification

The New York State Department of Health has just published a "Report on the Application of Ozone to Water Purification," by **Russell Spaulding,** Consulting Ozone Engineer, State Department of Health.

The report comprises 45 large pages and contains profusely illustrated notes on some of the European plants using ozone for the sterilization of drinking water as follows:

The plant at Saint Maur supplying the City of Paris daily with 24,300,000 gallons of sterilized water; bacteriological tests and ozone determinations in 1907 at Saint Maur plant; results of bacteriological examinations made on filtered water sterilized by ozone at the municipal plant of the City of Paris at Saint Maur on December 10, 1909; the ozone plant at Villefranche; mineral analysis and composition of the filtered water of the River Vesubie; the ozone plant at Nice; the ozone plant at Paderborn, Germany; the ozone plant at Saint Petersburgh, Russia.

The conclusions of the report are as follows:

"The records of the foregoing ozone purification plants, and their successful operation over an extended period of time, have established beyond cavil that ozone as a purifying agent is both efficient and economical, so that an extended discussion on the subject seems superfluous.

"It may be well, however, to emphasize the fact that in con-

tradistinction to all other purifying agents, ozone possesses one signal virtue in that it is absolutely non-toxic. Furthermore, it is insoluble in water to any appreciable extent, so that 'overdosing' is impossible.

"Aside from its wonderful and unfailing bactericidal power, ozone totally removes from water all traces of odor, color and taste due to the decomposition of organic matter. Ozonized water is never flat to the taste—it is not only perfectly hygienic, but satisfies the most delicate palate.

"To summarize, ozone destroys in water all that which is harmful or distasteful and in no manner changes the mineral qualities.

"The successful application of ozone to water purification is dependent on a number of factors, none of which may be disregarded with impunity.

"The failures that have occurred have been due solely to faulty design in the electrical or mechanical devices used.

"Ozone, as such, never has and never can fail to perform its functions properly.

"The first problem that confronted the pioneers in this art was the *economical production of ozone*.

"Ozone can be produced chemically, thermically and electrically. The first two methods were soon discarded in favor of electrical methods, which, however crude at the start, offered the best promise of ultimate success.

Energy Expended in Producing Ozone

"A molecule of ozone (O_3) weighs $16 \times 3 = 48$ grams.

"Its production requires a heat expenditure of 29,600 calories corresponding to $0.4235 \times 29{,}600 = 12{,}535$ kilogrammeters.

"To obtain 1 gram of ozone there is required an expenditure of energy of

$$0.4235 \times 29{,}600 \div 48 = 261 \text{ kilogrammeters.}$$

"Therefore, the theoretical value of 1 horsepower hour* (270,000 kilogrammeters) expressed in grams of ozone is

$$270{,}000 \div 261 = 1{,}034 \text{ grams}$$

"And the theoretical value of 1 kilowatt hour of electrical energy expressed in grams of ozone is

$$1{,}034 \times 1{,}000 \div 746 = 1{,}386 \text{ grams}$$

"The *efficiency of production of ozone* is even to-day quite remote from the theoretical efficiency. The early efforts produced only 3 grams of ozone per kilowatt hour of energy. The present specific output, prevalent in the European plants described in this report, varies from 40 grams to 60 grams of ozone per kilowatt-hour of electrical energy.

"In some instances much higher efficiencies have been obtained by supplementary devices, such as refrigerators and dehydration of air to be ozonized, but these devices are not only expensive and cumbersome but they tend to complicate the apparatus as a whole and diminish its dependability.

"Efficiency in ozone production has increased with the efficiency in electrical apparatus such as dynamos and transformers, but little progress has been made in increasing the efficiency of the ozone apparatus itself.

"Aside from the generators required to supply alternating current and the transformers used to step-up the potential to its required power, the ozone apparatus consists of electrodes of metal separated by an air space, dielectrics, such as glass, being interpolated between electrodes to produce a brush-discharge. The brush-discharge acting on the air disturbs the atomic balance of the oxygen, which is normally di-atomic, in such a manner that more or less of the atoms recombine in an unstable tri-atomic form which is called ozone.

"This brush-discharge is accompanied by the phenomena of light, heat, and noise, all of which absorb energy, thereby reducing the percentage of energy available for the creation of ozone.

"The heat generated tends to cause a reversion of ozone into oxygen and furthermore lessens the dielectric resistance of the glass dielectrics, making them liable to puncture by the high-tension current—thus putting the apparatus out of commission.

"To overcome this difficulty various means have been resorted to, to prevent a rise in temperature, such as refrigerating the air before it reaches the electrodes of discharge, or surrounding the electrodes with some fluid such as oil or water, for the purpose of carrying off the heat as fast as generated. These palliatives, though they make the operation of ozone generators possible, only add to the losses already entailed in the generation of the objectionable heat.

"The obvious remedy for low efficiency is, therefore, to so arrange the electrical circuit that the major portion of electrical energy supplied to an ozone generator shall be used in the conversion of oxygen into ozone, and that only a small portion shall be lost in a non-productive manner.

"The non-production of heat should be sought rather than its elimination by cooling means.

"To summarize then, the main desiderata in generating ozone are: 1, a supply of alternating electric current at low cost; 2, an efficient transformer to obtain high tension; 3, ozone electrodes that do not generate enough heat to (a) disrupt the dielectrics, (b) cause reversion of ozone to oxygen, (c) require external means for cooling; 4, ozone electrodes that will approach the theoretical efficiency much more closely than the various systems now in use.

"Such an ozone generator does exist and when offered to the public will revolutionize the art. Pending such time it would be improper to give a full description of it.

"The *efficiency in the application of ozone* to the purification of water is dependent entirely upon mechanical factors.

"Bacteria are immediately destroyed and converted into CO_2 when brought into contact with ozone. It becomes, therefore, necessary to insure this contact beyond possible failure. To do this the water must be broken up into very fine particles and this must occur in the presence of and in contact with ozone or ozonized air at a proper concentration.

"The only failures in ozone water-purification plants that have so far occurred have been due to a failure in breaking up the water in sufficiently fine particles and in properly contacting these small particles with ozone. I speak of failures in the technical sense, that is, failure to produce odorless, tasteless, colorless water, entirely free from pathogenic germs, by means of ozone.

"Failures might occur from a commercial point of view, because of excessive overhead charges or too high a cost of maintenance due to low efficiency or unreliability of apparatus.

"Two much-advertised plants on the American continent—one at Lindsay, Ontario, and the other at Ann Arbor, Mich.—have been failures. These failures are due only to faulty engineering in installation; there is an ample supply of ozone, but the water is not properly broken up and contacted with the ozone."

The author finaly feels "justified in urging upon the authorities of the state of New York the advisability of giving to ozone, as a means for purifying potable waters, their most earnest consideration.

"There is certainly no means known to science that is more reliable or absolute than ozone with which to overcome pollution and protect the public health.

"The latest developments in the art have brought ozone water-purification well within the boundaries of economical municipal administration."

*The report reads here horsepower instead of horsepower hour. As this is an evident mistake, it has been corrected here and in the following lines.—Editor.

Water Power Decision of the U. S. Supreme Court

A decision of immense and serious importance to water-power interests has just been handed down by the U. S. Supreme Court in the case of the United States, plaintiff in error, versus the Chandler-Dunbar Water-Power Company, et al. In view of the effect which this decision may have in discouraging in future the investment of large sums of money in the development of the water-powers of streams which now are non-navigable, but the water of which may be declared by Congress to be necessary for navigation through a system of canals some day in the future, we give a long abstract of the decision. The decision is printed as Senate Document No. 51.

By Act of Congress of March 3, 1909, the War Department was directed to take over all lands and property north of the present St. Mary's falls ship canal throughout its entire length and lying between the said ship canal and the international boundary line at Sault Ste. Marie, the act declaring that the ownership of such lands and property by the United States "is necessary for the purposes of navigation of said waters and the waters connected therewith." Provision was also made in the act for condemnation proceedings to determine compensation for the property taken, and for revocation of all permits to water-power companies on the lands specified.

Condemnation proceedings were accordingly instituted and the compensation finally awarded was as follows:

(a) To the Chandler-Dunbar Company, $652,332. Of this $550,000 was the estimated value of the water-power.

(b) To the St. Mary's Falls Power Company, $21,000.

(c) To the Edison-Sault Electric Company, $300,000, which has, however, been settled by stipulation.

(d) To the Michigan Lake Superior Power Company, nothing.

From these awards the Government, the Chandler-Dunbar Company, the St. Mary's Falls Power Company, and the Michigan Lake Superior Power Company had sued out writs of error.

The errors assigned by the United States challenged the allowance of any compensation whatever on account of any water-power right claimed by any of the owners of the condemned upland, and also the principles adopted by the District Court for the valuation of the upland taken. On the other hand, the several corporations who sued out writs of error, complained of the inadequacy of the award on account of water power claimed to have been taken, and also of the valuation placed upon the several parcels of upland condemned.

The opinion of the Supreme Court, written by Justice Lurton, then reads as follows:

"It will be seen that the controlling questions are, first, whether the Chandler-Dunbar Company has any private property in the water-power capacity of the rapids and falls of the St. Mary's River which has been 'taken' and for which compensation must be made under the fifth amendment to the Constitution; and, second, if so, what is the extent of its water-power right and how shall the compensation be measured?

"That compensation must be made for the upland taken is not disputable. The measure of compensation may in a degree turn upon the relation of that species of property to the alleged water-power rights claimed by the Chandler-Dunbar Company. We, therefore, pass for the present the errors assigned which concern the awards made for such upland. . . .

"The technical title of the Chandler-Dunbar Company includes the bed of the river opposite its upland on the bank to the middle thread of the stream, being the boundary line at that point between the United States and the Dominion of Canada. Over this bed flows about two-thirds of the volume of water constituting the falls and rapids of the St. Mary's River. By reason of that fact and the ownership of the shore the company's claim is that it is the owner of the river and of the inherent power in the falls and rapids, subject only to the public right of navigation.

"While not denying that this right of navigation is the dominating right, yet the claim is that the United States in the exercise of the power to regulate commerce may not exclude the rights of riparian owners to construct in the river and upon their own submerged lands such appliances as are necessary to control and use the current for commercial purposes, provided only that such structures do not impede or hinder navigation and that the flow of the stream is not so diminished as to leave less than every possible requirement of navigation, present and future.

"This claim of a proprietary right in the bed of the river and in the flow of the stream over that bed to the extent that such flow is in excess of the wants of navigation constitutes the ground upon which the company asserts that a necessary effect of the act of March 3, 1909, and of the judgment of condemnation in the court below, is a taking from it of a property right or interest of great value, for which, under the fifth amendment, compensation must be made. . . .

"This title of the owner of fast land upon the shore of a navigable river to the bed of the river is at best a qualified one. It is a title which inheres in the ownership of the shore, and, unless reserved or excluded by implication, passed with it as a shadow follows a substance, although capable of distinct ownership. It is subordinate to the public right of navigation, and however helpful in protecting the owner against the acts of third parties, is of no avail against the exercise of the great and absolute power of Congress over the improvements of navigable waters. That power of use and control comes from the power to regulate commerce between the States and with foreign nations. It includes navigation, and subjects every navigable river to the control of Congress.

"All means having some positive relation to the end in view which are not forbidden by some other provision of the Constitution are admissible. If, on the judgment of Congress, the use of the bottom of the river is proper for the purpose of placing therein structures in aid of navigation, it is not thereby taking private property for a public use, for the owner's title was in its very nature subject to that use in the interest of public navigation. If its judgment be that structures placed in the river and upon such submerged land are an obstruction or hindrance to the proper use of the river for purposes of navigation, it may require their removal and forbid the use of the bed of the river by the owner in any way which in its judgment is injurious to the dominant right of navigation. So, also, it may permit the construction and maintenance of tunnels under or bridges over the river, and may require the removal of every such structure placed there with or without its license, the element of contact out of the way, which it shall require to be removed or altered as an obstruction to navigation."

Various former opinions are quoted and the Supreme Court's decision continues as follows:

"The conclusion to be drawn is, that the question of whether the proper regulation of navigation of this river at the place in question required that no construction of any kind should be placed or continuued in the river by riparian owners, and whether the whole flow of the stream should be conserved for the use and safety of navigation are questions legislative in character; and when Congress determined, as it did by the act of March 3, 1909, that the whole river between the American bank and the International line, as well as all of the upland north of the present ship canal, throughout its entire length, was 'necessary for the purposes of navigation of said waters and the waters connected therewith,' that determination was conclusive.

"So much of the zone covered by this declaration as consisted of fast land upon the banks of the river, or in islands which were private property, is, of course, to be paid for. But the flow of the stream was in no sense private property, and there is no room for a judicial review of the judgment of Congress that the flow of the river is not in excess of any possible need of navigation, or for a determination that if in excess the riparian owners had any private property right in such which must be paid for if they have been excluded from the use of the same.

"That Congress did not act arbitrarily in determining that 'for the purposes of navigation of said waters and the waters connected therewith,' the whole flow of the stream should be devoted exclusively to that end, is most evident when we consider the character of this stream and its relation to the whole problem of lake navigation."

The decision thus gives a review of the situation of lake commerce and of the growth of canal traffic at the Soo. "Millions of public money have already been expended, in the construction of canals and locks, by this Government upon the American side, and by the Canadian Government upon its own side of the rapids, as a means by which water craft may pass

around the falls and rapids in the river. The commerce using these facilities has increased by leaps and bounds." . . .

"The upland belonging to the Chandler-Dunbar Company consists of a strip of land some 2,500 feet long and from 50 to 150 feet wide. It borders upon the river on one side, and on the Government canal strip on the other. Under permits from the Secretary of War, revocable at will, it placed in the rapids, in connection with its upland facilities, the necessary dams, dikes, and forebays for the purpose of controlling the current and using its power for commercial purposes, and has been for some years engaged in using and selling water power. What it did was by the revocable permission of the Secretary of War, and every such permit or license was revoked by the act of 1909. . . .

"That it did not thereby require any right to maintain these constructions in the river longer than the Government should continue the license needs no argument. They were placed in the river under a permit which the company knew was likely to be revoked at any time. There is nothing in the fact which favors the estopped in law or equity. The suggestion by counsel that the act of 1909 contemplates that the owner should be compensated not only for its tangible property, movable or real, but for its loss and damage by the discontinuance of the company's license and its exclusion from the right to use the water power inherent in the falls and rapids, for commercial purposes, is without merit.

"The provisions of the act in respect of compensation apply only to compensation for such 'property described' as shall be held private property taken for public uses. Unless, therefore, the water-power rights asserted by the Chandler-Dunbar Company are determined to be private property the court below was not authorized to award compensation for such rights.

"It is a little difficult to understand the basis for the claim that in appropriating the upland bordering upon this stretch of water, the Government not only takes the land but also the great water power which potentially exists in the river. The broad claim that the water power of the stream is appurtenant to the bank owned by it, and not dependent upon ownership of the soil over which the river flows has been advanced. But whether this private right to the use of the flow of the water and flow of the stream be based upon the qualified title which the company had to the bed of the river over which it flows or the ownership of land bordering upon the river, is of no prime importance. In neither event can there be said to arise any ownership of the river. Ownership of a private stream wholly upon the lands of an individual is conceivable; but that the running water in a great navigable stream is capable of private ownership is inconceivable.

"Whatever substantial private property rights exist in the flow of the stream must come from some right which that company has to construct and maintain such works in the river, such as dams, walls, dikes, etc., essential to the utilization of the power of the stream for commercial purposes." . . .

"That riparian owners upon whom public navigable rivers have in addition to the rights common to the public certain rights to the use and enjoyment of the stream, which are incident to such ownership of the bank, must be conceded. These additional rights are not dependent upon title to the soil over which the river flows, but are incident to ownership upon the bank. Among these rights of use and enjoyment is the right, as against other riparian owners, to have the stream come to them substantially in its natural state, both in quantity and quality. They have also the right of access to deep water, and when not forbidden by public law may construct for this purpose wharves, docks, and piers in the shallow water of the shore. But every such structure in the water of a navigable river is subordinate to the navigation, and subject to the obligation to suffer the consequences of the improvement of navigation, and must be removed if Congress in the assertion of its power over navigation shall determine that their continuance is detrimental to the public interest in the navigation of the river." . . .

"It is said that the 12th section of the act of 1909 authorizes the Secretary of War to lease, upon terms agreed upon, any excess of water power which results from the conservation of the flow of the river and the works which the Government may construct. This, it is said, is a taking of private property for commercial uses and not for the improvement of navigation. But, aside from the exclusive public purpose declared by the eleventh section of the act, the twelfth section declares that the conservation of the flow of the river is 'primarily for the benefit of navigation and incidentally for the purpose of having the water-power developed, either for the direct use of the United States or by lease . . . through the Secretary of War.'

"If the primary purpose is legitimate, we can see no sound objection to leasing any excess of power over the needs of the Government." . . .

"The conclusion, therefore, is that the court below erred in awarding $550,000, or any other sum, for the value of what is called 'raw water,' that is, the present money value of the rapids and falls to the Chandler-Dunbar Company as riparian owners of the shore and appurtenant submerged land."

The decision then takes up the award for the upland taken.

The awards to the Chandler-Dunbar Company includes certain sums for special values: The value of the upland strip fixed at $65,000 was arrived at in this manner:

(a) For its value, including railroad side tracks, buildings and cable terminals, including also its use, "wholly disconnected with power development or public improvement, that is to say, for all general purposes, like residences, or hotels, factory sites, disconnected with water power, etc., $20,000."

(b) For use as factory site in connection with the development of 6500 hp, either as a single site or for several factories to use the surplus of 6500 hp not now used in the city, an additional value of $20,000.

(c) For use for canal purposes an additional value of $25,000.

The United States excepted to the additional value allowed in consequence of the availability of land in connection with the water power supposed to be the property of the Chandler-Dunbar Company, and supposed to have been taken by the Government in this case. It also excepted to so much of the awards as constituted an additional value by reason of availability for locks and canals.

"These exceptions so far as they complain of the additional value to be attached to these parcels for use as factory sites in connection with the development of horsepower by the Chandler-Dunbar Company, must be sustained. These 'additional' values are based upon the erroneous hypothesis that that company had a private property interest in the water power of the river, not possibly needed now or in the future for purposes of navigation, and that that excess or surplus water was capable, by some extension of their works already in the river, of producing 6500 hp.

Having decided that the Chandler-Dunbar Company as riparian owners had no such vested property right in the water power inherent in the falls and rapids of the river, and no right to place in the river the works essential to any practical use of the flow of the river, the Government cannot be justly required to pay for an element of value which did not inhere in these parcels as upland. . . .

"The requirement of the fifth amendment is satisfied when the owner is paid for what is taken from him. The question is what has the owner lost, and not what has the taker gained." . . .

"The exception taken to the inclusion as an element of value of the availability of these parcels of land for lock and canal purposes must be overruled. That this land had a prospective value for the purpose of constructing a canal and lock parallel with those in use had passed beyond the region of the purely conjectural or speculative. This land was the only land available for the purpose. It included all the land between the canals in use and the bank of the river. Although it is not proper to estimate land condemned for public purposes by the public necessities or its worth to the public for such purpose, it is proper to consider the fact that the property is so situated that it will probably be desired and available for such a purpose."

The Relative Importance of Principles and Practice in Education.*

By Dr. James Douglas

My first visit to Colorado was made in 1872. My mission was to report on the California mine in Gilpin county, which had been sold to a Scotch company, and was being restored to the original owners by other methods than those of bargain and sale. As a mine expert I got a lesson in how *not* to conduct legitimate mining. On the other hand I saw the first successful Western attempt at smelting argentiferous and auriferous copper ores at the old Black Hawk works of Senator Hill; and I made the acquaintance of a brilliant young man, Richard Pearce, in charge of the Swansea works, at the bend of Clear Creek, below Georgetown. This was only forty years ago, but the changes which have been made in metallurgy since then suggest many reflections on methods of scientific education, and on the equally important habits in self-training we should cultivate and practice on ourselves.

Early Metallurgical Practice

Let us glance at these changes. The Boston and Colorado smelter, under the management of Professor Hill, who had been a professor at Brown University, commenced operations in 1867 with two hearth roasting furnaces and two reverberatories, 15 ft. by 9 ft., heated by wood as fuel at $5 per cord. The lump sulphides were calcined in heaps. Each reverberatory received its charge of two tons of roasted ore and roasted concentrates, which constituted a self-smelting mixture and yielded a matte of about 40 per cent copper. This was shipped to England for refining and separation.

But it was Mr. Pearce to whom the credit is due for taking the lead in enlarging the reverberatories, increasing their capacity and reducing the fuel consumption. In his address as President of the American Mining Institute of Mining Engineers in 1889,[1] he tells us that "the furnaces at Argo, to which place the works were moved from Black Hawk, had been constructed with a capacity of twenty-five to twenty-seven tons per day, and with a fuel consumption of one to three." The following table, supplied me by him for my Cantor Lectures,[2] illustrates the steady growth of the reverberatory:

Year	Size of furnace ft. in.	Size of furnace ft in.	Area of hearth, sq. ft.	Area of fire box, sq. ft.	Ratio of hearth to fire box, sq. ft.	Tons cold ore smelted in 24 hrs.	Tons hot ore smelted in 24 hrs.	Tons coal consumed in 24 hrs.
1878	15 0 x	9 8	108	22.55	4,800:1	12	..	5
1882	17 10 x	10 4	143	22.50	6,351:1	17	..	7
1887	21 2 x	12 8	202	24.75	8,161:1	24	..	9
1891	24 4 x	14 2	262	28.50	9,192:1	28	..	10
1893	30 0 x	16 0	390	32.50	12,000:1	35	43	13
1894	35 0 x	16 0	456	32.50	14,030:1	..	50	13.5

But startling as was that progress for that day and generation, it was slow in comparison with that of late years, not so much by increasing the size of the reverberator as by improving its operation.

Development of the Reverberatory

Mr. E. P. Mathewson, in his paper before the recent eighth International Congress of Applied Chemistry, on the "Development of the Reverberatory," says:

"The next step in development was made in Butte, Montana, by the Colorado Smelting Company, this plant being at that time affiliated with the Argo works, so that Mr. Pearce's influence was apparent. The step referred to was the lengthening of the hearth to 50 feet, with consequent increase in capacity to 105 tons in 24 hours.

"The first furnace of this size built from the Colorado Smelting Company's plans, was constructed at the Butte and Boston plant in Butte, in the year 1900."

At Cananea, under Dr. Ricketts, the use of oil as fuel and the recovery of waste heat were studied with care. But the highest results seem to have been attained at the plant of the Steptoe Company at McGill, Nevada, with California oil as fuel.

Mathewson says: "A record performance at McGill, communicated by Superintendent Sorenson on December 17th, 1911, No. 1 furnace smelting 660 tons of total charge on an oil consumption of five-eighths of a barrel of oil per ton of charge.

Steptoe Plant at McGill, Nevada.
Record Tonnage on December 17th, 1911.
Record of Running of No. 1 Reverberatory Furnace, and Analysis of Charge.

Total charge per furnace day, tons.....	666.
Oil fired per furnace day, bbl.....	421.
Coal equivalent of oil fired, tons.....	124.0
Total charge per bbl. of oil tons.....	1.58
Oil, bbl. per ton of total charge.....	.63
Equivalent gross coal as % of total charge.....	18.60"

The substitution of oil for coal has, from the point of view of cheaper operation and the control of the heat, taken place wherever the difference in the cost of the unit of heat value does not forbid it. The low cost of smelting now attained in the reverberatory is attributable also to the conversion of the waste heat, as it escapes from the throat of the furnace, into steam. This loss has always been recognized, though with the old type of boilers it was never found practicable to remedy it; but in this, as in most other metallurgical processes, the mechanic has co-operated with the metallurgist to consummate what each alone would have failed to accomplish. From forty-five to fifty-five per cent of the heat generated by the fuel is recovered as steam. The combination of the reverberatory with the blast furnace, instead of the often unreasonable exclusive preference for the one over the other, is now finding many advocates. At the Cooper Queen works at Douglas we use both, and there we have imitated Cananea in fettling our reverberatory furnaces by pouring in ore of suitable fusibility along the wall of the furnaces through openings in the roof.

Influence of Bessemer Converting and Electrolytic Refining

But the development of the reverberatory would not have revolutionized the smelting and refining of auriferous and argentiferous copper ores had not two other great inventions intervened during the short period of time under our review. I refer to the introduction of the pneumatic method, through the Bessemer converter, and the refining of copper by electrolysis.

It was about 1882 that the news of M. Manhes' successful application of the converter to the concentration of copper induced Mr. Franklin Farrel to introduce the pneumatic method into his Parrot works in Butte. Difficulties, of course, beset him, but they were so slight that by 1885 he had six stands in successful operation. Since then, the principle and main features of practice being accepted, the progress, both in concentrating to bullion and incidentally in learning how to smelt crude ore in the converter, has been such that the converter has become more essential to the copper metallurgist than to the steel worker. It has grown in size from 4½ feet in diameter to 20 feet, and from a capacity of one and a half tons of matte per charge to thirty-three tons.

Instead of an acid lining requiring frequent replacement, the shell is now lined with basic bricks so refractory that, under proper precautions, 15,000 tons of copper are poured from a single lining, this first lining being still in use; and owing to the acid ingredients of the charge a much wider latitude of ores can be allowed for the acid ingredients of the charge, necessary to eliminate the iron from the matte, than when special clays and quartz had to be selected on account of their plastic or other qualities.

The converter has thus become a smelting furnace as well as an appliance to eliminate the older methods of concentration

*Commencement Address, Colorado School of Mines, May 23, 1913.
[1] *Transactions*, A.I.M.E., Vol. xviii, 1889.
[2] Canton Lectures before the Society of Arts, London.

by repeated roastings and smeltings, which involved so much fuel and time and labor.

And almost simultaneously with the practical introduction of the converter, improvements in the generator to produce electric current in large quantities and cheaply, made possible the electrolytic refining of copper and the separation of all impurities, including gold and silver.

First Electrolytic Refining of Copper

When I was superintendent of the Chemical Copper Company's works at Phoenixville, in 1878, Mr. Franklin Farrel, to whom the copper industry owes more, perhaps, than to any one man of the vanishing generation, induced me to make an experiment in the electrolysis of copper, at first on a small scale, with battery current, and then on a working scale, with generators, and in vats arranged in series and in multiple arc.

Our ignorance was replaced by the knowledge and gracious assistance of Mr. Edward Weston of Newark, who loaned me three of his nickel-plating dynamos, and, better still, came to Phoenixville more than once to advise and help me out of difficulties. I suppose I may claim the merit of making, in this country, the first electrolytic copper by the ton, but the merit is really due him, who in this and innumerable other instances has concealed his disinterested work for his favorite science and pursuits under a thick veil of modesty and generosity.

My idea at first was that the cathode might be rolled into coherent sheets. I soon discovered my mistake, but the copper was all bought at high figures by the Shaker community, who found that the cathodes yielded an extraordinarily good quality of metal for anodes in their depositing vats. The Chemical Copper Company was always in difficulty. It started with a shortage of capital, and never covered the shortage. The works came to an ignominious close. The company was not prosperous. The price of copper in 1878 and 1879 was at a low ebb, but took a sudden jump to twenty-five cents before the close of 1879. I had a nice stock of copper on hand and in the vats and was doing well; but the temptation to realize on such a soaring market was irresistible, and I was ordered to refine everything I could find, at the sacrifice of even the precious metal contents.

Excuse this personal digression. It was one of the blind steps in the progress of events which had a momentous influence on your immediate neighborhood; for the rapidity of the pneumatic process and the perfection of the electrolytic process inevitably supplanted the more intricate, though more interesting, process from a metallurgical standpoint, Pearce's modification of the Ziervogel method, as practised first at Black Hawk and then at Argo.

Prices Paid for Ore by the Boston & Colorado Smelter

Up to the date of Mr. Pearce's alliance with Senator Hill the mattes, as I have said, were shipped to England. And the prices paid for ore and concentrates were low. Fortunately, as a compensation, the value of silver was high. Mr. Pearce, in his presidential address to the American Institute of Mining Engineers in 1889, gives the proportion of values paid the miners in those early days. He says:

"The following table, which I have prepared from data collected by a friend, will show the commercial advantages which the miner has experienced by the progressive development of the smelting industry. For the sake of comparison, I have selected ores which have no special value as fluxes, or as aids to smelting, and will consider them merely in relation to their intrinsic value, and endeavor to show the returns which the miner gets now, as compared with the figures of eighteen years ago. In other words, the net percentage value of an ore to the miner today is compared with the value for the successive periods from 1871.

"The value of the silver has been figured from $1.29 per ounce in 1871, down to 93 cents in 1899; and for the years prior to the resumption of specie payment the premium on gold is taken into consideration. The slight falling off in 1874 and 1875 was due to the depression following the financial panic in the fall of 1873. For a time silver ores were rather a drug on the market, and prices consequently fell off.

Table Showing Percentage of the Total Value of Ores Paid to Miners Each Year During a Period of Eighteen Years.

Place	Year	Contents in silver per ton	Percentage of total value paid to the miner
Black Hawk.....	1871	100 oz.	65
	1872	100 oz.	65
	1873	100 oz.	65.5
	1874	100 oz.	53.6
	1875	100 oz.	60
	1876	100 oz.	67.2
	1877	100 oz.	64.3
	1878	100 oz.	65
Argo.....	1879	100 oz.	70
	1880	100 oz.	74
	1881	100 oz.	74
	1882	100 oz.	76
	1883	100 oz.	76.5
	1884	100 oz.	81
	1885	100 oz.	77
	1886	100 oz.	80
	1887	100 oz.	80
	1888	100 oz.	82
	1889	100 oz.	84

"The difference between the maximum and minimum is 30.4 per cent.

"I have avoided making any figures to show the changes in value of gold-ores from year to year, but, going into details, I may state that a Gilpin county gold-ore which would net the miner 53 per cent of its value in 1870 would now yield him 80 per cent, a difference of 27 per cent; and on a somewhat lower grade of ore the difference is 33 per cent."

Since then the gold would yield the miner 90 per cent. Comparing the prices paid for concentrates in 1874 and 1889 he quotes 24 per cent of the value as that of 1874 and 78 per cent as that in 1889.

Professor Hill in 1872 defended himself in the Central City *Register* against the accusation of extortionate charges based upon unjust comparison of his furnace treatment with California milling practice. The professor says:

"The Boston and Colorado Smelting Company are treating ores, of which the gross value of the gold and silver, estimated in currency, is $50, $100, and $150, at a cost to the miner of $35, $40, and $45 respectively; that is, for ores which contain $50 per ton, currency value, all over $35 is paid to the seller, and for ores containing $100 per ton, also all over $40 is paid to the seller, and so on. For intermediate grades a *pro rata* charge is made.

"This company also pays for the copper $1.50 for each per cent on the dry Cornish assay, which is the assay on which all copper ores are sold.

"No one who is acquainted with the facts will deny that the ores of Colorado are the most complex which are worked on this continent, containing, as they do, mixed with the sulphurets of antimony, arsenic, zinc, and lead, and a refractory gangue. Neither can any one deny that the actual costs of all the principal elements employed in smelting, viz., fuel, labor, firebricks, and iron, are more than double here what they are east of the Mississippi River, and much higher than they are in California."[3]

Dr. Raymond, in his report as Commissioner, for 1870,[4] gives the following as the scale of prices paid at the Hill works:

"For ore containing per ton—	Is paid of the value
2 ounces gold.....	20 per cent
3 " "	30 " "
4 " "	40 " "
5 " "	45 " "
6 " "	50 " "

"For silver, seventy-five cents per ounce is paid, after deducting as many ounces of silver as there are per cent of copper in the ore. For copper $2 for each per cent, deducting one-half per cent from the amount indicated by wet assay.

[3]Mineral Resources West of the Rocky Mountains, 1872, R. W. Raymond.
[4]Ibid., 1870.

No account is taken of quantities less than one ounce of silver, one per cent of copper, or one-quarter ounce of gold. The above rates are in coin."

Early History of Smelting in Colorado

In 1875 Professor Egleston published a paper on the Boston and Colorado Smelting Works,[8] after Mr. Pearce had joined Professor Hill and was separating the precious metal. His description is significant of the small scale on which work was still conducted, and the high terms on which ore was bought. He says:

"The works are thus located in the very center of the gold and silver producing regions of Colorado, and are also most favorably situated with regard to transportation. They treated in 1874 30 tons of ore and tailings in 24 hours, and produced 700,000 ounces of silver, 12,000 ounces to 15,000 ounces of gold, and 225 tons of copper. With matte from Alma their production in 1875 will be 110,000 ounces of silver, 25,000 ounces of gold, and 250 tons of copper.

"The gold ores are divided into three classes. The first class consists of auriferous copper pyrites containing from 2 to 10 per cent of copper, 2 ounces to 10 ounces of gold, and 2 ounces to 10 ounces of silver. These ores average 4 per cent of copper, 3½ ounces of gold, and 6 ounces of silver. The second class are tailings from the gold mills, consisting of pyrites with about 1½ per cent of copper, 1¼ ounces of gold, and four ounces of silver. The third class consists of tellurium ores; which have a very silicious gangue, and contain 100 ounces to 200 ounces of gold, and 6 ounces to 10 ounces of silver. These ores come mostly from Boulder county, and are often worth $10,000 to $15,000 to the ton.

"The silver ores of the first class consist of surface ores, mostly free from sulphur, containing 70 per cent of silica. They contain 100 ounces of silver and 5 to 6 per cent of lead, and no gold. Those of the second class are sulphurets, rich in blende and poor in galena and pyrites; they contain 150 ounces of silver, 15 per cent of zinc and lead, and no gold."

The next contribution to the literature of our subject was the presidential address from which I have quoted.

By that time the smelting industry of Colorado had expanded from about 20,000 tons per year in 1877 to 596,594 tons in 1888, and Argo was treating some 200 tons of copper-bearing material from all sections of the Rocky Mountains, including the matte from the company's famous branch works in Butte, operated under the superintendence of Mr. Williams.

When Mr. Pearce had introduced the Ziervogel method of separating copper and silver, substantially as practised in Germany and England, and his own modification of parting the gold, the heavier expense of shipping the matte to England or Germany was avoided, and, as the above table showed, the miners shared in the saving.

But the method was delicate, and costly, involving not only skill but many operations. And consequently it has not been able to compete with matte concentration in the converter, and the perfect separation of the precious from the baser metal in the electrolytic vat. The requiem of this famous enterprise was thus pronounced[9] in "Mineral Resources for 1910," Part I, Page 196:

"The year witnessed the dismantling of the plant of the Boston and Colorado Smelting Co. and the passing of this pioneer in the copper smelting industry of the West. Although this company has been successful in its long period of operations, the management did not consider it wise to rebuild the plant, which was becoming out of date. This pioneer plant has had a long and successful career, but copper smelting plants constructed in the State in recent years have been far less successful and have been able to operate but intermittently."

The moral in its effect on us as students is to regard methods, even the most ingenious and scientifically perfect, as merely stepping stones across the river of industrial life, the further side of which is still far away.

[8] *Transactions*, A.I.M.E., Vol. iv, 1875.
[9] Mineral Resources of the United States, 1910, Part I, page 196.

Pre-eminent Value of Principles in Education

Looking back, therefore, on the changes in these two branches of mining and metallurgy, namely, in the improvements and changes of methods and the higher price which the miner receives today for the products of his industry, we are forced to recognize how, in our system of education, the study of principles should occupy a more prominent place than the study of practice. Even our acceptance of principles, outside of mathematics, should be held as open to modification, but in the main the fundamental laws of physics and chemistry may be accepted, as we interpret them, to be correct and our only safe guides.

Every improvement made in your own region has been through a better understanding of these laws, and carrying them into operation.

For instance, the effect of the injection of air into molten metal was perfectly understood before Bessemer brought mechanical ingenuity as well as chemical science to bear upon the solution of the pneumatic method. The practicability of extracting the carbon to the exact point at which pig iron is converted into steel proved to be so difficult that the Bessemer process would probably have had a limited range of usefulness had not the suggestion of another chemist been adopted, to oxidize all the carbon, then add to the charge a specific amount of carbide of manganese with a known quantity of carbon to re-carbonize, while the manganese absorbed any dissolved oxide of iron. The whole success of this momentous improvement, which was not discovery at all, depended upon the application of known facts to meet certain practical conditions.

Following along the line of steel manufacture, the adoption of the basic lining in the converter, following the Thomas-Gilchrist proposal to line the open-hearth furnace with a basic material to eliminate the phosphorus was a simple application of known chemical facts.

Development of the Copper Converter

Turning from the pneumatic method as applied to iron to similar methods as applied to copper, we see Holloway carrying his smelting of ore and concentration of matte up to the point where metallic copper began to form; then the occlusion of the tuyeres in the bottom of the steel converter by the chilled copper; and the loss of years in the adaptation of this simple method to the concentration of copper till M. Manhes, adopting a type of converter with elevated tuyeres, which Mr. Bessemer had himself patented, brought the tuyeres within reach of a punching bar, and thus substantially revolutionized the metallurgy of sulphides of copper.

The wonderful progress in the study of the generation of electricity, its transmission, its conversion into power, and its electrical action has been made within the short period we have been reviewing. It is a special branch of study and research, the intricacies of which the average student cannot thoroughly master, but with the general laws of which he should be familiar. The carrying out of any mining scheme which utilizes electricity involves the employment of an expert in the person of an electrical engineer, but if you are to be an efficient manager of the mine, mill or furnace plant you must personally know what can be done with this mighty but mysterious force, though you may have to leave the handling of it to your electrician.

Another element which of late we have called to our use for service at a distance from the point of its generation is compressed air. To determine whether in each particular case transmission by electricity or by compressed air or by a combination of both should be adopted involves on the part of the manager a fair acquaintance with the fundamental laws of electricity and pneumatics, if he is to be the controlling factor in the administration, instead of a puppet in the hands of his subordinate officers. He may not be familiar with all the practical appliances or the latest improvements for applying these principles, but he must be sufficiently master of the subject to be the master of his staff.

Value of Industrial Chemistry in Development of the West

There is another field which the mining and metallurgical engineer of the future will have to cultivate, and that is industrial chemistry. It is, of course, a branch of industry in which co-operation between the chemical manufacturer and the miner and metallurgist must be closer and keener than it is at present. In fact, our metallurgical activity has been greater than the chemical, partly due to the fact that the market for chemical products has been widely distant from the possible sources of some of the more necessary chemical ingredients. But this distance is being gradually eliminated.

One of the most important iron and steel works of the country is already established at Pueblo, and there is no reason why within a limited period important chemical industries should not travel westward with the westward wave of population. Then the demand for sulphuric acid and that immensely valuable and yet noxious gas, sulphurous acid, which we are pouring into the atmosphere all over the West, will come into urgent demand.

But I think that the miner and metallurgist should not wait. He should himself take the lead in developing a Western chemical industry. We are wasting the basic ingredient—sulphuric acid. You have within easy reach probably the largest deposits of phosphate of lime rock in the world. The farmer will need superphosphates and it rests with you to supply him, and in doing so develop new mining industries and recover a valuable by-product.

Problems in Metallurgy of Rare Elements

In addition, your State possesses probably more rare minerals than you have any idea of, the recovery of which and the extraction of their precious contents will require of you a closer study of chemical operations than has been usually demanded of the students of metallurgy. Take, for instance, your carnotite ores. The separation of the three valuable elements—radium, uranium, and vanadium—should become immediately a branch of metallurgy.

In addition to the sulphurous acid which now goes to waste from your furnace stacks, the waste gases from your coke ovens should pave your roads with asphalt and fertilize your fields with sulphate of ammonia. It is evident, therefore, that the alliance of mining and metallurgy and the chemical industry must become in the future much closer than it has been in the past.

And to get that total out of the material we treat, or if not the total more than has heretofore been recovered, will be one of the tasks of the future. The decline in the value of the ores of the Gilpin, Cripple Creek and other districts is reflected in the falling off of the mineral production of the State.

This decline must be compensated for, not only by reducing the losses in treatment of the standard elements recovered, but by saving others now wasted. Looking over the table of your mineral products, I see that this process is already in operation, for zinc appears for the first time in the list in 1902, and the value of this once neglected element saved in 1912, $8,591,623, was greater than that of your silver, $5,023,960; and it alone enables you to maintain your position in the ranks of producing States.

But there are other valuable ingredients—let us call them by-products—which might be utilized, and to do so the assistance of the industrial chemist must be secured. You, as supplying the material on which he works, must know more of technical chemistry than has heretofore been thought necessary.

Fundamental Investigations Required for Future Prosperity

We are living at the close of a period of almost unnatural activity and prosperity. We have had half the new world to exploit; and, with steam and electricity as our aids, in little more than a generation we have depleted the fertility of the soil and emptied our mines of their richest ore, taking from each the stores nature had accumulated nearest the surface. The harvest has been abundant and cheaply garnered; but a future generation cannot reap crops without planting, and in our case, if mining prosperity is to be maintained, it must be by saving every product and by-product, and widening, therefore, the field of our operations and consequently of our studies.

The experience here of the past generation demonstrates how rapidly practice has changed under the application of scientific principles. The conclusion, therefore, is that, as a fundamental maxim of education, our studies should aim at a better understanding of the deep and broad laws which underlie all practice rather than the ephemeral methods of the day, and thus train our minds in habits of original research rather than obediently and servilely following the practice of the past or even of the present. The study of methods, however, cannot be neglected; but, if familiar with the principles, you are better able to use your principles in assisting you to devise methods for carrying them into practice.

The danger of too much knowledge of principles is that the scholar becomes so conscious of his shortcomings in practice and sees so keenly the possible perfection of practice ahead that he does not devote himself with sufficient energy to working out and practising the imperfect, practical methods and using tools he has at his hand to work with. Some of the cleverest men I have had to do with have developed this fault. It should be your endeavor to familiarize yourselves with the scientific basis of the subjects you have to deal with without destroying your practical ingenuity and being depressed by the evident shortcoming of your best efforts to carry principles into action.

Electro-Analysis of the Copper Alloys

By J. G. Fairchild

In the analysis of brass and bronze, a common method of procedure consists in the oxidation and removal of tin and antimony by means of nitric acid, followed by their solution in ammonium hydrosulphide. This solution, after separation of the small amounts of the sulphides of copper, lead and iron, is then electrolyzed with the addition of potassium cyanide as reducing agent. The strong tendency of this alkaline solution to form polysulphides prevents the complete deposition of the metals. Although this treatment is apt to fail, yet the necessary operations are short. This is an important factor in a commercial laboratory. The old method of purifying the oxides of tin and antimony by fusion with sulphur and soda, with the subsequent weighing of the stannic oxide after antimony has been removed, appears to be too long to admit of practical use. Furthermore, to gravimetrically determine tin is liable to result too high and is recommended only when dealing with traces. With the solution of this first problem, the remaining successive operations offer little difficulty.

The alloy is dissolved in 15 cc. nitric acid, 1:1, and evaporated nearly to dryness, or until the liquid becomes a thick syrup. It is convenient to use a beaker of 250 cc. capacity. Addition is made of 5 cc. more acid with 50 cc. of water and the solution is boiled for a few minutes. After filtering, the stannic oxide is washed well with a hot acid solution of ammonium nitrate. A convenient strength is 2 per cent of the salt with 2 per cent of nitric acid. This wash aids in freeing the tin from copper, besides preventing the tendency of the precipitate to be carried through the paper.

The precipitate is washed back into the beaker and dissolved in five grains each of ammonium oxalate and oxalic acid*. The solution is diluted to 100 cc. and boiled till clear.

The filter paper is next placed in the solution and disintegrated by boiling, the solution is refiltered and the paper pulp is washed with hot water. When tin is under 10 per cent, as in most brasses, it is needless to purify the solution.

Several color determinations of copper occluded by this amount of tin have shown usually less than 1/10 of 1 per cent. Since copper is the greatest impurity lead and iron count for still less.

*One grain of NH_4Cl must be added here to aid in the deposition of tin.

If it is desired to make a separation of this copper, or if antimony is present, heat the solution to about 95 deg. C. and treat with H_2S for forty-five minutes according to G. W. Thompson's method.

The tin is filtered from the antimony and copper sulphides which are washed well with hot water. The precipitate is washed back into the beaker with a little water and treated with 2 grams of KOH.

The liquid is warmed till the sulphides blacken, when all the antimony will be in solution. Add 50 cc. concentrated HCl to the antimony in a 200 cc. Erlenmeyer flask, together with one gram of $KClO_3$, and boil till the sulphur is oxidized and the ClO_2 is all expelled.

While still warm add one gram of KI to the solution, cool quickly in ice water with further addition of cold water and titrate the antimony with thiosulphate.

The small amounts of copper and lead are dissolved in a little strong HNO_3, evaporated to near dryness and added to the main copper solution.

Tin.—Boil the filtrate from the copper and antimony to expel H_2S and transfer hot to a dish of 100 cc. capacity. The surface of this dish should be 1 square decimeter. A nickel dish can conveniently replace the usual one of platinum. Use a current density of 0.8 amp per square decimeter and pass a slow stream of air through the solution.

In order to prevent loss by bursting of the air bubbles, it is convenient to pass the stem of the spiral anode up through a small inverted glass funnel. The glass tip through which passes the compressed air is inserted beneath the funnel. It is also safer to further cover the dish with a pair of split watch glasses. The air circulation need be only two bubbles per second.

Deposition is complete in about 2½ hours, as indicated by a cessation of the vigorous gasing of the electrolyte. At this point alkalinity begins, as shown by a litmus paper test. The electrolyte is siphoned off as usual and the dish, after rinsing with alcohol, is ignited and dried over a Bunsen flame. The deposit of tin made in this manner is dense and brilliant.

Phosphorus.—The electrolyte from the tin determination contains all the phosphorus, which is oxidized by evaporation with nitric acid and precipitated by ammonium molybdate as usual.

Copper.—This metal may be deposited in three hours by using the same current density as above. Lead is obtained simultaneously on the spiral. The weight of peroxide for most brasses is less than 20 milligrams—an equivalent of 1.73 per cent metal on the one gram sample. If the lead is so large in amount that it falls off the spiral and adheres to the copper, dissolve both metals and deposit the lead first with reversed poles.

Zinc.—The electrolyte from the copper and lead determinations is evaporated rapidly over a Bunsen burner. Use a large porcelain casserole and bake the residue on a hot plate till ammonium nitrate is completely volatilized. This salt must be expelled before the deposition of zinc. When the residue is thoroughly dry, cool and introduce 1 cc. concentrated H_2SO_4 with a little water. Heat till the zinc and iron are completely dissolved. Dilute to 50 cc. with the addition of 5 grams NaOH and a few grains of Na_2O_2. The zinc readily dissolves as sodium zincate.

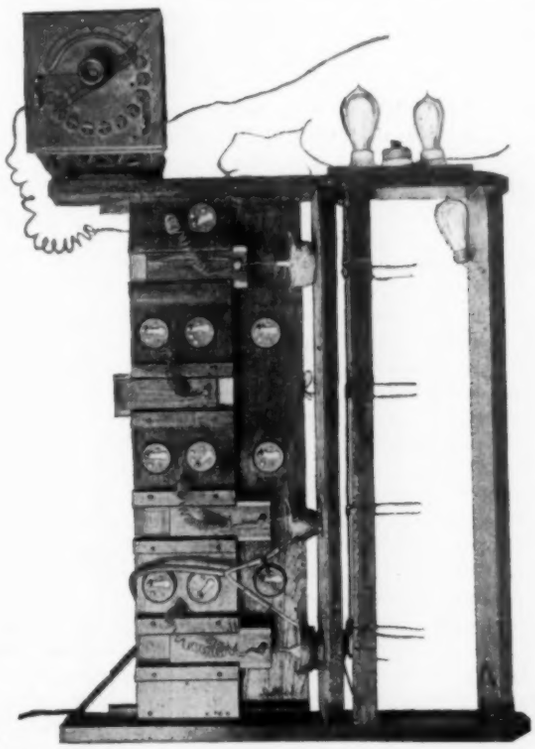
After heating the solution for a few minutes, filter through a plug of asbestos and glass wool inserted in a glass funnel for the removal of iron, etc. If the zinc amounts to more than 20 per cent take half of the solution for electrolysis. More NaOH is added to make the strength 10 per cent, and the electrolyte is heated to about 60 deg. C. By using the copper-coated platinum dish for zinc an extra weighing is dispensed with.

Zinc is completely deposited in from two to two and one-half hours. This point is determined by withdrawing 2 cc. or 3 cc. in a test tube, boiling out the dissolved oxygen and adding an excess of NH_4HS. After a thorough warming and shaking the solution should remain clear throughout its depth. Aluminium gives the same reaction here as zinc, so re-examine the electrolyte if a cloud should appear.

Iron.—Dissolve the residue on the asbestos with dilute HCl. This small amount of iron is conveniently and accurately estimated by reduction with KI and titration with thiosulphate. Copper, if not already completely removed, will interfere with this method for iron.

Manganese.—This will be found with the iron. The two oxides are dissolved in hot nitric acid, 1:1, by the aid of bisulphite. The solution is divided into two equal parts. One is evaporated to SO_3 fumes with addition of some H_2SO_4, after which HCl is added and titration made as before. The other is oxidized with sodium bismuthate and the manganese titrated as in iron and steel.

Nickel.—This metal, if present, can be determined after the titration of iron. The solution is reduced with SO_2, made alkaline with NH_4OH, then acid with acetic acid and treated



ELECTROLYTIC BOARD AND WIRING

with H_2S while warm. Small amounts of NiS are best weighed as NiO.

Aluminium.—This metal may be looked for in the electrolyte from the zinc and determined as usual.

Suggested Procedure When Dealing with White Metals

No better scheme can be recommended than that of G. W. Thompson, with one or two exceptions. According to his procedure, the alloy is dissolved in aqua regia, lead chloride crystallized out with alcohol and the filtrate evaporated to dryness. It is then made alkaline with KOH and oxidized with H_2O_2. A large excess of ammonium oxalate and oxalic acid is next added, and the antimony, copper and traces of lead separated by means of H_2S. The tin is electro-deposited.

The antimony is separated from the copper and lead by solution in KOH, then oxidized with HCl and $KClO_3$ and titrated. Copper and lead are dissolved in nitric acid and electrolyzed. Any iron will be weighed with the tin.

Zinc, if present, appears as insoluble zinc oxalate which is filtered from the soluble tin and converted to sodium zincate for electrolysis.

An exception is made to his procedure in that lead chloride is converted to nitrate and electrolyzed. Use 15 cc. HNO_3 in the volume of 100 and 0.25 amp per square decimeter.

The initial temperature of the electrolyte should be about 60 deg. C. in order to prevent the formation of metallic lead on the cathode. Two hours is sufficient for 150 milligrams of peroxide. During the last half hour the current density is increased to 0.5 amp.

Notes and Precautions.

For the determination of copper, an oval-shaped crucible of the same metal and of 100 cc. capacity will be found very convenient and inexpensive. In order to prevent any attack of the crucible by the acid electrolyte, connections are made before the solution is introduced.

Chromic acid is an excellent addition agent for copper, in that it prevents the formation of nitrous acid.

So long as the last deposit remains bright and adherent the successive layers need not be removed.

The outside of this crucible should be gold plated in order to prevent oxidation of the copper. Plating of the inside is optional.

Zinc is also weighed in the same crucible.

A containing vessel of this shape affords better circulation of the electrolyte.

When starting the action of the compressed air, dip the glass tip into a beaker of water; otherwise some of the electrolyte might be blown out by too sudden a rush of the air.

In determining tin in white metals, it is better to so partition the solution as to provide for not more than 0.2 gram for electrolysis. This amount can be deposited in three hours, but addition of more oxalic acid to the electrolyte may be necessary.

The largest amount deposited in this laboratory has been 0.3 gram.

If a check determination is wanted on a high tin electrolytically deposited, a good method consists in the solution of the drillings in a flask provided with a Bunsen valve. A few pieces of marble are then added and the flask cooled in ice water. The tip of a burette is furnished with a delivery tube which is passed into the flask. By this means the stannous chloride can be sucked up into the burette in order to avoid oxidation as a result of pouring it through the air. Delivery is then made into a weighed amount of boiling ferric chloride solution and the excess of tin is titrated back with iodine.

When dealing with high antimony, such as found in type metal, Mohr's method is recommended. This involves solution with HCl and $HClO_3$, reduction with KI and titration of the liberated iodine with thiosulphate.

A good air circulation is essential in all the electrolytic determinations, especially so in those of lead and zinc. The safe limit of deposit for 1 square diameter of surface has been found to be 150 milligrams of lead peroxide.

In converting zinc to zincate, be sure that solution is complete, both in H_2SO_4 and subsequently in NaOH. Wash the asbestos plug with 10 per cent NaOH. Not more than 0.2 gram has been deposited thus far, but this limit may possibly be increased by an addition agent, such as glue. This colloid has the property of making the zinc crystals smaller and hence more adherent. It also offers an obstruction to the vigorous evolution of oxygen gas from the anode, and thereby lessens the danger of mechanical loss of metal from the electrolyte.

By determining the copper, lead and tin in two dishes at the same time, a bronze analysis may be completed in about one day.

Caustic potash is much less efficient than the soda for solution for zinc, as the latter forms a more soluble compound with zinc than the former.

With a view of eliminating the evaporation to dryness of the copper electrolyte containing the zinc, the suggestion is made to apply the lactic acid electrolyte as given in Smith's Electrochemical Analysis.

I wish to credit Mr. F. W. Dugan with the design and construction of the electrolysis board and wiring.

Laboratories of Dr. Chas. F. McKenna,
New York City.

The Modern By-Product Coke Oven

One of the most interesting papers read at the recent meeting of the American Iron and Steel Institute was presented by **C. A. Meissner,** Chairman of the Coke Committee of the United States Steel Corporation, on the modern by-product coke oven. The complete paper comprises 64 pages and is very full and complete. On account of its length only a summary of its contents may be given here. One special section of the paper which should be of particular interest to our readers will be printed in full in our next issue.

The author first gives coke and coal statistics of the world and of Germany and the United States. The United States Steel Corporation in 1912 used a total of 40,877,862 tons of coal. For coking purposes, the Steel Corporation used 24,401,577 tons of coal. They produced a total of 16,719,387 tons of coke, of which 5,164,547 tons were made in by-product ovens and 11,554,840 tons were made in bee-hive ovens.

After giving the principal historical figures of the development of the coke industry, the author takes up the advantages of by-product coke ovens over bee-hive coke ovens and makes, concerning the cost, the following interesting remarks. They have studied the comparative cost of by-product coke ovens versus bee-hive coke ovens very carefully and find that this depends entirely on the location of the bee-hive oven plant and the attendant conditions of such a location. Comparing, for instance, a by-product plant at Gary, Ind., with a bee-hive plant in the Pocahontas, West Virginia, region, both on a large scale, and taking into consideration the mine investment, houses required for workmen and everything connected with the construction ready for operation of either type, they find that the bee-hive oven installation complete costs more than the by-product oven installation complete for the same number of tons coke produced per year. Comparing the above by-product plant with a similar bee-hive oven installation in the Connellsville region, where all conditions are more favorable and where the difference in the yield of coke from coal coked in the bee-hive oven and in the by-product oven is very materially less than it is in the Pocahontas region, they find that the cost of a bee-hive oven installation complete is considerably less than the cost of a by-product oven installation complete, per ton coke produced per year.

A feature of supreme importance in the use of the by-product oven is the greater coke yield obtained in this type than in the bee-hive oven from the same coals. This amounts to about 23 per cent. to 25 per cent. in the low volatile coals, such as Pocahontas, and from 5 per cent. to 12 per cent. in the high volatile coals, dependent on their original content of volatile matter. This subject is not fully understood and generally appreciated and yet is of vast importance in the conservation of our coal fields. A concrete example will be of interest:

	Pocahontas Coal Coked in	
	Bee-Hive Coke Oven Plant.	By-Product Coke Oven Plant.
Number of ovens.....	6,154	560
Coking time	72 hours	17½ hours
Yield of coal to coke.....	60%	82%
Net tons of coal required to produce 2,880,000 tons coke per year.....	4,800,000	3,512,000
Net tons of coal saved per year by use of by-product coke ovens for above coke production	.....	1,288,000

This amount of coal saved, if it were coked in by-product ovens, would produce about 1,000,000 tons of coke per year.

Then follow coal data, a discussion of the quality of by-product coke for blast furnace use, construction features of by-product coke ovens, notes on coke quenching and screening, illustrated descriptions of various types of coke ovens (Semet-Solvay, Koppers, Otto, Collin, Coppee, Still), and a discussion of by-product recovery operation and apparatus.

Progress of Electric Smelting at Heroult, California

An Authoritative Report of the Progress Made at the First American Electric Iron Ore Reduction Plant. By John Crawford, Plant Manager

The plant of the Noble Electric Steel Company is situated on the banks of the Pitt River at Heroult, Shasta County, California, a station on the Sacramento Valley & Eastern Railroad, which branches off the Southern Pacific Railroad at a station called Pitt, lying about six miles to the west of Heroult. The plant consists of one 2000-kw and one 3000-kw iron furnace of the long and narrow type, one 2000-kw iron furnace of the shaft type (not now operated), one 2000-kw steel furnace of the tilting type (not now in operation), twenty charcoal retorts of the horizontal type of 2¼-cord capacity, a refinery to handle the by-products of the wood distillation, and four 5-ton unit lime kilns

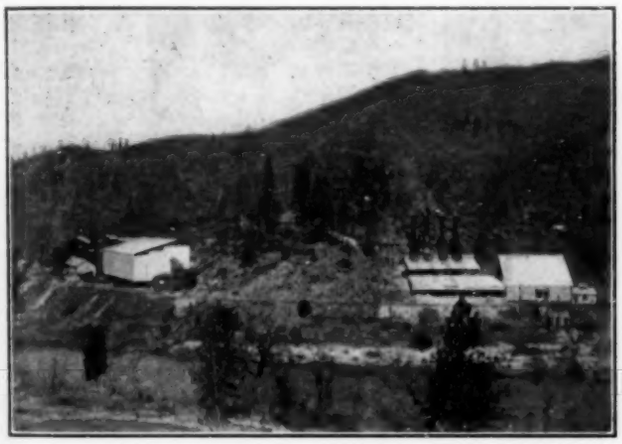


FIG. 1—VIEW OF HEROULT LIME KILNS TO RIGHT, FURNACE BUILDINGS TO LEFT

of the continuous type. Beside these there is a power substation and the necessary machine shops, office, residence, and furnace buildings, etc.

Introduction

To build an electric furnace for making pig iron which can be operated as a technical success is quite an achievement, but this having been accomplished, there still remains the very considerable problem of making it an economic success.

With approximately the same costs for power, charcoal, and stock, local conditions have caused us to abandon the type of furnace which we are told in Sweden they are operating commercially.

If a certain type of furnace is found to be best suited to make a grade of iron for which the demand is limited, either a market must be found for this product, and that market educated to accept it, or else the design of the furnace, or method of operation must be altered so as to make a grade of iron for which there is a market already established. In Sweden, conditions permitted them to adopt the former and simpler course, while we at Heroult have had to resort to the latter.

The principal users of pig iron on the Pacific Slope and Far Western states are custom foundries. Specialty foundries, such as those making stoves, bath tubs, pipe, etc., are still relatively few, as are also open-hearth steel furnaces, so that to operate electric furnaces successfully on the Pacific Coast, grades of iron must be produced which meet the demands of general foundry purposes.

There is on the coast an abundance of scrap cast iron, and foundries making what are spoken of in the East as heavy castings are few in number; hence the popular demand is for a soft high-silicon iron which is a good scrap carrier and can be easily machined when in light castings.

Thus it is apparent our problem became one not merely of making pig iron successfully but of making iron with a silicon content of from 2 to 3 per cent economically. When a blast furnace works "off" it ordinarily means only a slight concession in price to get rid of the low-grade iron, while with us here if iron runs under 1 per cent silicon it means a large concession in price to dispose of it. As electric furnaces like blast furnaces have the faculty of misbehaving at times conditions imposed on us the necessity of having a furnace which would respond readily to alterations in the furnace burden and still be of large enough capacity to be efficient.

History

It may be of interest to run briefly over the experiments conducted at Heroult which have led up to our present furnaces. For some twenty-five years one of the stockholders of this company has controlled a deposit of magnetite of considerable extent which is exceptionally high-grade and runs very low in sulphur and phosphorus.

The following analysis represents the average of several scarfs taken across a quarry face 70 ft. wide and 45 ft. high. The quarry face has now been extended to a width of 115 ft. and is still in ore.

Analysis of Magnetite

SiO_2	3.43	per cent
Al_2O_3	0.81	" "
CaO	0.70	" "
MgO	0.32	" "
MnO	0.28	" "
CuO	Trace	
Fe_3O_4	79.63	" "
Fe_2O_3	14.56	" "
	99.73	per cent
Fe	67.86	" "
P	0.0116	" "
S	0.021	" "

Geologically the deposit has been formed by deposition from ascending waters and lies between the quartz—augite—diorite of this region and a metamorphosed (McCloud) limestone. The analysis of this limestone which is also very pure is as follows:

SiO_2	1.02	per cent
Al_2O_3	0.61	" "
MgO	1.12	" "
CaO	53.80	" "
FeO	0.20	" "
CO_2 (by diff.)	43.25	" "
	100.00	per cent

In 1906 these properties were brought to attention of H. H. Noble, president of the Northern California Power Company, who, after some correspondence with Dr. Paul L. T. Héroult, built in 1907, a 1500-kw furnace of the three-phase resistance type. Pig iron was produced in this furnace, but mechanically it was weak.

In May, 1908, there was designed by E. E. Cox and others under the management of D. A. Lyon, a furnace of the shaft type embodying the principle of a shaft terminating at its base in a bosh which was super-imposed on an arched crucible through which the electrodes penetrated at an angle. This furnace which was put in operation in the summer of 1909, by a singular coincidence is almost identical with the furnace built at Trollhättan, Sweden, about the same time. As descriptions

FIG. 2—FURNACE BUILDING

of the Swedish furnace have appeared so frequently in technical journals, I believe the design of this type of furnace is generally familiar, so I will omit a detailed account.

This furnace was operated intermittently until the spring of 1911. As chemist at the plant during part of this time, I was enabled to observe the mechanical and metallurgical workings of the furnace closely and was imbued with the idea that it could be made economically efficient.

But from the style of its construction, it is apparent that it could not be made to respond readily to changes in the burden, and in order to make consistently high-grade foundry iron this is an essential. In a blast furnace, too great an excess of coke in the burden can be taken care of by increasing the quantity of air blown in, but in an electric furnace this is, of course, not feasible, as the oxygen would attack the electrodes. The excess of carbon must be taken care of by an excess of oxygen put into the furnace through increasing the ore in the burden.

The necessary excess one might think could be calculated to a nicety, but because of unknown factors practically the excess must be added gradually until the controls on the slag from the iron and the general working of the furnace show the desired result to have been accomplished. This sometimes takes several days.

If the excess of carbon has been allowed to proceed too long, the furnace will, of course, "freeze up." The slag will give up part of its lime content to form calcium carbide, part of its alumina to form aluminium carbide, part of its silica to form silicon carbides, and part to form ferrosilicon, and part of the carbon remaining turns to beautiful sparkling flakes of graphite. Remarkable examples of molecular replacement of the carbon in the charcoal by silicon carbide has also been noted, making beautiful specimens of petrified charcoal. I once saw a furnace present these phenomena, though there is no excuse for it, as the decrease in the amount of stock going into the furnace and the daily controls on the slag and metal should give ample warning. The matter of too little carbon gives less trouble, and if the furnace is producing low silicon and carbon iron should give none at all.

The question may be fairly put: Why cannot the necessary carbon in the burden be calculated within sufficiently close limits to prohibit any possibility of trouble? Theoretically, of course, it can be. By daily analyses of the furnace gases taken at regular distances as they ascend from the crucible up the stack the ratio in which the carbon is actually being oxidized to CO and CO_2 can be approximately determined, and this together with the controls on the slag and metal will keep the carbon within safe limits if the furnace is running with a low carbon burden; that is, making low-silicon iron.

When, however, the furnace is running on a high carbon burden, calculated to make a 3 per cent silicon iron, it is apparent that the carbon must be carefully regulated. Insufficient carbon not only lowers the grade of the iron, but introduces difficulty by throwing an excess of SiO_2 into the slag. Too much carbon lowers the efficiency of the furnace besides leading to the difficulties mentioned above.

Practically, it is always necessary to carry an excess of carbon over theoretical calculations to take care of the atmospheric oxygen and moisture which is occluded in the charcoal and the atmospheric air which is inevitably drawn into the furnace after it has been in use in spite of every practical precaution to prevent it. It is evident, of course, that these are to a considerable extent unknown factors; hence the charcoal must be continually varied, as the control analysis indicates that it is too high or too low.

So, while the shaft type of furnace gave promise of economic success, it operated on the low-silicon, low-carbon, white iron (which our Swedish friends have dignified by the name of pig steel), it was shut down for making foundry iron.

Following this, an attempt was made under the direction of S. T. Wellman to work out the theory of reducing the ore with the proper charcoal and flux to sponge iron in a series of small, portable electric furnaces which were then picked up one at a time by a mechanical charging machine like those used in charging open-hearth steel furnaces and their contents emptied into an electric smelting furnace similar in design to the arc-type of tilting steel furnaces. Mechanical difficulties and inability to control the metallurgy of the process caused it to be abandoned after a very brief period of operation.

General Description of the Present Furnaces

Subsequent to this, R. E. Frickey and others after considering the causes of failure of the previous furnaces designed a

FIG. 3—ELECTRIC PIG IRON

2000-kw three-phase furnace of the long and narrow type having four electrodes delta-connected and suspended between five charging stacks. This is the type of furnace which we are now operating and which seems best adapted to our needs.

The first furnace was put in operation in November, 1911, and has been working since, except for periods when operations were suspended to effect advantageous mechanical and electrical changes. All was not smooth sailing, of course.

Pioneer work never is, but we became satisfied that this type of furnace promised economic success and a companion furnace of 3000 kw capacity was built.

This type of furnace consists of a rectangular steel shell, 28 ft. long and 10 ft. wide, lined with standard furnace brick. The lower part of the furnace is battered four ways and corresponds to the blast furnace crucible, while the upper part of the furnace corresponds to the bosh or smelting zone of the blast furnace. The furnace is surmounted by five charging stacks, 18 ft. high, which are battered so as to prevent the descending burden from hanging up. Between the stacks the top of the furnace is arched over and through the center of these arches the electrodes penetrate vertically into the charge.

The electrodes used are of Acheson graphite, 12 in. in diameter, and are connected with a tapered male and female joint.

Up until now, no arrangements have been made on these furnaces to utilize the furnace gases as the way in which to utilize them best had to be studied rather as an economic than a metallurgical problem; that is, whether the saving in charcoal effected by circulating the gases back through the furnace and taking advantage of the reducing action of the CO is greater than the saving in fuel by burning them under lime kilns, char-

FIG. 4—CHARGING RETORT—SHOWING TRANSFER TROLLEY CAR

coal retorts, or elsewhere. Our observations have led us to adopt the latter course.

As to the claims made for the cooling effect of the circulated gases on the furnace roof, I have never considered this worthy of practical consideration on a resistance-type furnace. If the charge is descending regularly it will protect the roof and if it is not, the slight lowering of temperature caused by the cool gases will not prevent the heat radiated from the electrode from melting out the roof.

Electrical Description

Power is supplied by the Northern California Power Company three-phase at 60,000 volts into our own substation. Here are six Westinghouse 1500-kw oil-immersed and water-cooled lowering transformers wound for a voltage ratio of 38,100 to 2200 volts, Y-connected on the high-tension side. The high-tension leads pass through a General Electric oil switch by which the furnace load is cut in or out.

The three 750-kilovolt-ampere furnace service transformers which set close up to the rear side of the furnace furnish three-phase currents at from 40 to 80 volts. They are specially designed General Electric transformers, oil-immersed and water-cooled and capable of standing an overload of 25 per cent. On the 2200-volt primary side are brought out eight current taps for voltage regulation. These are taken to an oil-immersed individual solenoid-operated contactor panel which is connected with the switchboard to give fifteen steps for voltage regulation.

The connections from each of the transformers to the electrodes on the low-tension side consist of eight pieces of ⅜-in. by 8-in. copper bus-bar connected in delta from the three transformers to the four electrodes.

The electrode holders are water-cooled cylindrical stuffing boxes made of 98 per cent copper. The electrodes are sus-

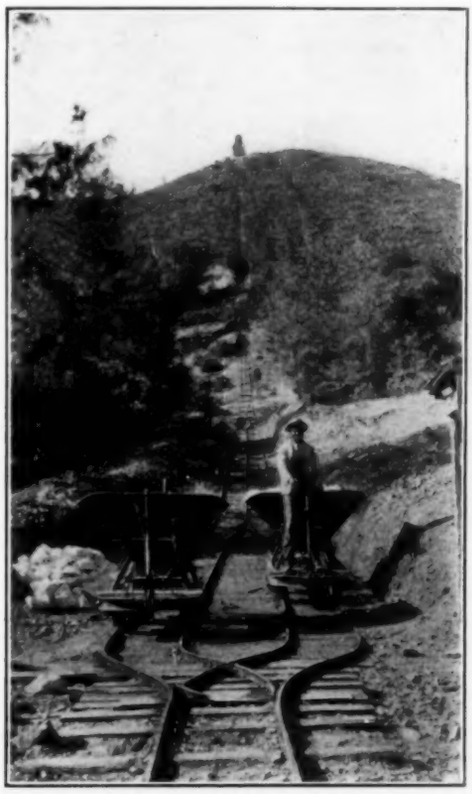


FIG. 5—FOOT OF GRAVITY TRAM

pended in these from above and the annular space between the electrode and the holder is packed with a specially prepared graphite capable of being compressed to a density equal to that of the electrode itself. This packing material offers no more resistance to the passage of the current than the electrode and because of its unctuous nature permits the electrode to be raised and lowered without breaking electrical contact.

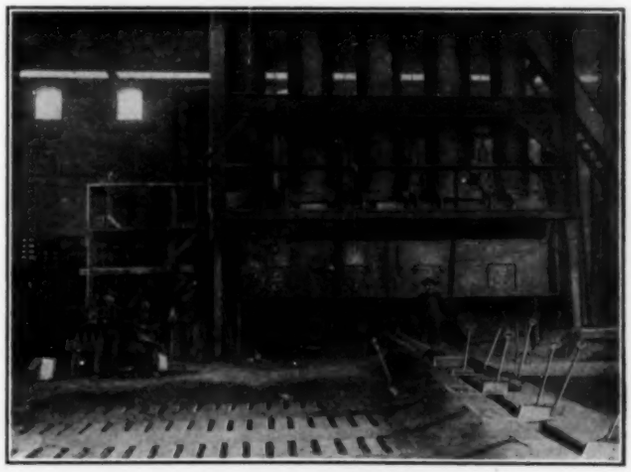


FIG. 6—FURNACE NO. 6

Electric control is through a switchboard on which are mounted, in rows of three, a separate set of meters for each phase. A set consists of a Thompson ammeter, a Thompson voltmeter, a Thompson wattmeter, and a Weston power-factor meter. Beneath the last named are three hand wheels to con-

trol the voltage variation and below these are three knife switches to control the load on the board. A separate recording wattmeter for each phase is also set on this board.

For operating the voltage control and the circuit-breaker, there is a 7½-kw motor-generator set comprising a 125-volt direct-current generator, direct connected to a 10-hp induction motor. Above these on a panel are mounted a circuit-breaker, an ammeter, voltmeter, and the necessary switches.

From an electrical standpoint, this type of furnace has worked far more smoothly than any with which I am familiar. When everything is normal inside of the furnace the instruments will show little variation throughout the twenty-four hours, except that the power-factors which average respectively 90, 85 and 70 per cent, improve a few points after the furnace is tapped and gradually fall off again as the molten iron accumulates in the bottom of the crucible.

Metallurgy

The stock is charged into the five charging stacks previously mentioned on the basis of 500-lb. units of iron ore. This small charging unit while it entails extra labor on the feed floor has the advantage of mixing the ore, charcoal, and flux as intimately as if the charge were bedded and homogeneity of charge is very essential.

With this type of electric furnace at least the ore is reduced to a very much greater extent by actual contact with the carbon than by the action of the CO in the stack gases. Some examples of gas analyses will bear this out.

Gas Analyses

CO_2	O	CO	CH_4	H
8.6	0.2	57.2	16.0	0.8
9.2	1.4	56.2	12.2	1.2
7.8	0.15	57.1	....	...
6.0	0.20	67.9	....	...
8.1	0.10	64.8	....	...
4.2	0.25	60.7	....	...
5.0	0.40	66.6	....	...

From these analyses it will be noted that the ratio of CO to CO_2 is very high, but by utilizing the calorific value of the gas,

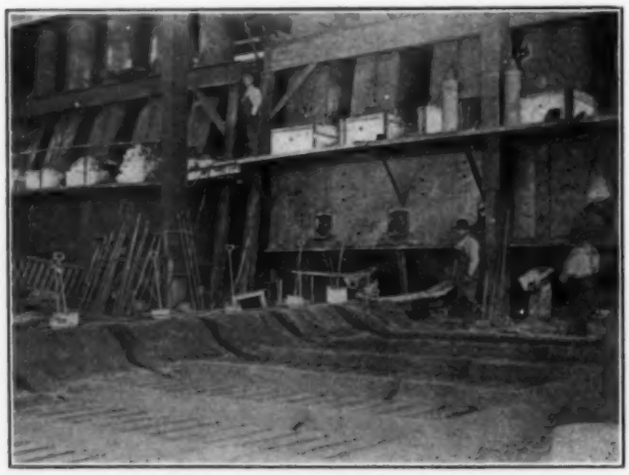


FIG. 7—NOS. 6 AND 7 FURNACES; PIG BED BEFORE NO. 7 FACE

the high charcoal consumption will be balanced to a very appreciable extent.

I have experimented to some extent with the possibility of utilizing coke or gas carbon instead of charcoal.

But the 72-hour metallurgical coke such as I have tried offers two objections: First, its electrical conductivity is so good that much of the current passes between the electrodes in the upper part of the furnace. The smelting zone is thereby raised and the furnace runs hot on top with attendant melting of the arches and cold at the bottom.

Second: This coke because, of its density and high crushing strain does not break down like charcoal as the burden descends; hence, less surfaces of carbon are exposed to be oxidized by the ore and there is a less intimate mixture of the two. Reduction of the ore takes place more slowly, the silicon in the iron is lowered, the power consumption per ton increases, and thus the efficiency of the furnace is reduced. However, by adopting certain precautions in crushing the stock and feeding same into the furnace I have operated on a mixture of 60 per cent coke and 40 per cent charcoal with a very fair degree of furnace efficiency and the grade of the iron was kept up to No. 2 foundry.

The possibilities of operating electric iron furnaces with coke

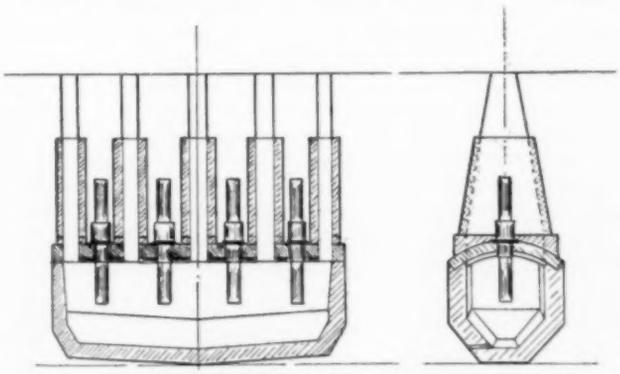


FIG. 8—SECTIONS OF NO. 7 FURNACE

instead of charcoal seem to me to offer a very interesting and necessary field for investigation. At present successful operation of electric iron furnaces depends among other things on an abundant and fairly cheap supply of charcoal. This generally limits their field of activity to well-timbered regions which are usually isolated and where freight rates are high. I believe that many of our coals which make a very poor metallurgical coke for blast furnace use on account of their low crushing strain might be found to make a satisfactory fuel for electric furnace use.

The fact that our ore is of so high a grade renders our metallurgical problem somewhat different from that usually encountered. The ore as it comes from the quarry will often run for weeks at a time as low as 2½ per cent SiO_2, and from 67 to 68 per cent Fe; so that to make pig iron which will run from 2 to 3 per cent Si, it is necessary to augment the silica in the ore by the addition of barren quartz. The requisite amount of lime or limestone is added so as to give theoretically a slag running about 47 per cent SiO_2. The following slag analysis will indicate the extent of the silica variation:

SiO_2	Al_2O_3	FeO	CaO	MgO
54.00	29.70	1.30	12.98	1.17
50.20	28.60	2.40	16.03	2.36
46.13	27.20	0.65	23.10	3.31

Ordinarily slags containing more than 50 per cent SiO_2 are too viscous to run well, but our slags are fluid up to 54 per cent SiO_2. This may be due to the high alumina ratio. As we only make from 125 to 140 lb. of slag per ton of pig, it is found more economical to permit a small percentage of the iron to escape in the slag than to attempt to reduce it. The depth of the green color in the slag also serves as a rough indication for the furnace-men as to whether the carbon ratio of the burden is becoming too high or too low.

In calculating the charcoal for the burden we assume all the carbon burns to CO, as we find it necessary any way (for reasons previously given) to carry an excess of charcoal in order to make high silicon iron. Thus to make 2.75 silicon iron the theoretical quantity of charcoal (containing 85 per cent fixed carbon) necessary is 35 per cent of the pig, whereas there is actually used about 40 per cent. Inasmuch as any necessary change in the burden is distributed over five stacks the furnace responds very rapidly and practically permits of the silicon in the iron to be controlled within a limit of 0.5 per cent.

The iron is tapped three times a day into sand pig-beds. A sample is taken from each bed from a pig 10 inches long cast for

the purpose. The drillings from the several beds are ground together for analysis. The sow and pigs are lifted from the beds by a traveling crane with grab-hooks and broken by a drop used in connection with an electric magnet.

Our system of grading is somewhat like that used in grading the Southern iron, except that we grade entirely by silicon content as the sulphur and phosphorus run uniformly under 0.04 per cent when we operate on charcoal alone.

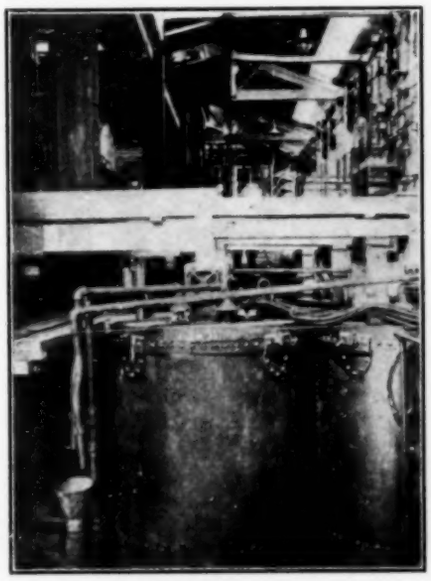


FIG. 9—SHOWING TRANSFORMERS AND LOW-TENSION TERMINALS (BUS BARS)

The system adopted by the American Foundrymen's Association for sampling carload lots is followed and we sell if necessary on a guaranteed silicon content within 0.25 per cent limits. Our grade card is as follows:

No. 1	"	Low	2.50	2.75
No. 2	"	High	2.25	2.50
No. 2	"	Low	2.00	2.25
No. 3	"	High	1.75	2.00
No. 3	"	Low	1.50	1.75
No. 4	"	High	1.25	1.50
No. 4	"	Low	1.00	1.25

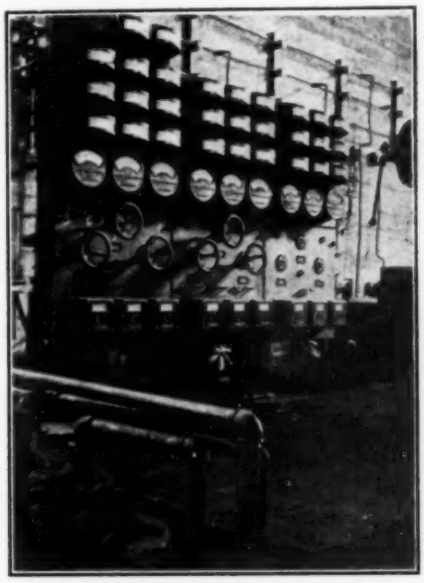


FIG. 10—ELECTRICAL INSTRUMENT PANEL

Though we generally limit our production to the four foundry grades.

I made some experiments during the early operations of this furnace to determine the relative variation of the silicon and carbon in the iron according as the carbon content of the bur-

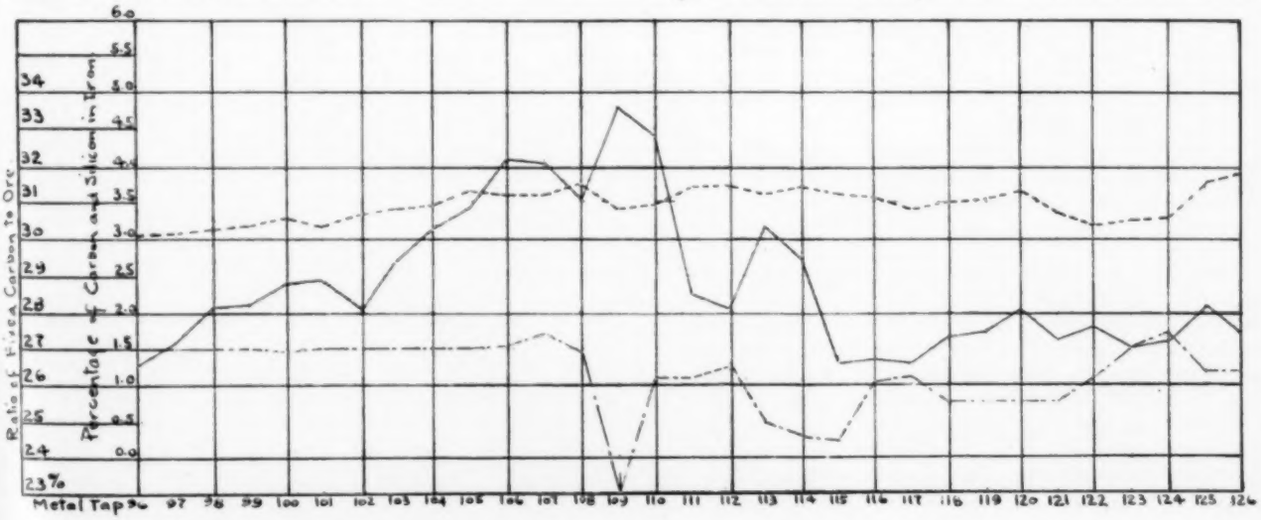


FIG. 11—VARIATION OF SILICON AND CARBON

	Silicon	Per Cent
No. 1 Silvery	4.5	5.0
No. 2 "	4.0	4.5
No. 1 Soft	3.5	4.00
No. 2 "	3.0	3.5
No. 1 Foundry High	2.75	3.00

den was increased or decreased. A portion of these results has been plotted graphically in Fig. 11. Each number represents a metal cap made at regular eight-hour intervals. It will be noted that while the silicon responds very readily to changes in the carbon ratio of the burden the carbon in the metal as would be expected, shows little variation. While the

temperature of the metal was not determined the same load was carried throughout so that approximately the same temperature existed after the first thirty-six hours. The silica in the slag is not shown on this plot, but it naturally rose and fell inversely with the silicon in the iron.

Since it is made with charcoal one would not expect the fracture of electric furnace iron to be as coarse as that of coke iron of equal silicon content, but the fracture of our iron is much finer and more uniform than that of charcoal blast furnace iron of the same grade. A very noticeable characteristic is the homogeneity of the fracture and the almost entire absence of segregations and "hard spots." It is also distinguished by its toughness, so that we have to cast it with very deep notches.

This iron has been used for making almost every article which comes within the province of the foundryman, and we have yet to hear a complaint where it has been used intelligently. The softer irons have already won a reputation for themselves as softeners and scrap carriers. Some analyses taken at random of carload lots of high, medium, and low grade iron will illustrate how constant all the other metalloids are except the silicon.

Analyses on a 200-ton lot shipped to a foundry for making steel castings and sold on a guarantee of from 2.75 to 3 per cent silicon and a maximum of 0.04 per cent of sulphur and 0.04 per cent of phosphorus:

Silicon	2.88 per cent
Combined carbon	0.09 " "
Graphite carbon	3.38 " "
Sulphur	0.028 " "
Phosphorus	0.031 " "

Analysis on a 100-ton lot shipped for general foundry purposes—silicon guaranteed from 2.25 to 2.50 per cent:

Silicon	2.42 per cent
Combined carbon	0.27 " "
Graphite carbon	2.94 " "
Sulphur	0.036 " "
Phosphorus	0.023 " "

Analysis on a 100-ton lot shipped to stove works to be mixed with high-phosphorus iron for stove castings. Sold on guarantee of from 1.75 to 2.00 per cent silicon and 0.04 per cent maximum of phosphorus:

Silicon	1.78 per cent
Combined carbon	0.60 " "
Graphite carbon	2.86 " "
Sulphur	0.028 " "
Phosphorus	0.030 " "

Furnace Efficiency and Conclusion

The efficiency of this type of furnace increases in slightly more than direct ratio to the increase in load, and it is of interest to note that as in the case of the electric steel furnaces the faster the furnace is operated the cooler the walls and roof are, and the smoother it works generally. I offer in explanation the theory that the faster smelting takes place the faster the cool charge can descend to protect the arches and walls.

As for technical efficiency, the long and narrow type of furnace is not the equal of the shaft type carrying the same load, though we have kept our power consumption as low as 2200 kw-hours per ton of pig when carrying a load of 3000 kw, but the long and narrow type offers a possibility of building several furnace units onto each other like copper blast furnaces. By this means the radiation losses and electrical losses will be cut down and the efficiency of what are in single-unit furnaces the end-electrodes will be greatly increased.

A bank of three or four units arranged in this way will admit of as simple metallurgical control as a single-unit and will have an efficiency, I believe, equal to a shaft type of furnace carrying the same load.

Further, it can be arranged so that part of the furnace can be frozen up and repaired while the remainder is operating, and for this reason its yearly output should exceed the shaft type.

From the nature of its construction it is capable of being made more nearly fool-proof metallurgically, mechanically, and electrically than the shaft type, and further the electrode consumption is lower, due to the fact that the electrodes penetrate vertically into the charge.

While I hardly agree with the prophecies made by some that electric furnaces for producing pig iron will eventually be competitors of blast furnaces even in the regions where economic conditions make the latter possible, I do feel that where electric power can be obtained cheaply and where coke and freight rates are high and for making superior grades of iron, electric reduction furnaces will enable many large bodies of iron ore to be worked which would otherwise remain idle and that the electric iron furnace both of the shaft type and of the long and narrow type, each in the field best adapted for it, will make steady progress.

Heroult, Shasta County, California.

British Iron and Steel Institute

The autumn meeting of the (British) Iron and Steel Institute will be held in Brussels, on Monday, Tuesday, Wednesday, and Thursday, September 1st to 4th, 1913.

The provisional program of the meeting is as follows:

Sunday, August 31. Secretaries' office open at the Palais des Academies, Brussels, for the registration of members' names and the issue of badges and programs.

Monday, September 1. Opening meeting in the Hall of the Palais des Academies. A selection of papers will be read and discussed. In the afternoon visits will be made to places of interest in Brussels. In the evening a reception will be held by the Burgomaster at the Hotel de Ville.

Tuesday, September 2. Meeting in the morning for the reading and discussion of papers, at the Palais des Academies. Afternoon visits to Colonial Museum and the Parc de Tervueren. In the evening his majesty King Albert will receive the members at the Royal Palace, Brussels.

Wednesday, September 3. A special train will leave in the morning for Ghent where a visit will be paid to the International Exhibition now being held in that city.

Thursday, September 4. Alternative excursions will be made to Liege and Charleroi. The Liege excursion will include a visit to the works of Messrs. John Cockerill & Co., Seraing, to the works of the Ougree-Marihaye Company, and to the Coppee Coke Oven Gas Plant at Athus-Grivegnee. The excursion to Charleroi will embrace visits to various metallurgical, glass and other works in the vicinity of that town.

The British members will travel on a special train, leaving Charing Cross on the afternoon of Saturday, August 30, at about 2.30 p. m.

Dr. Arthur Cooper is the president, and Mr. G. C. Lloyd, 28 Victoria St., London, S. W., is the secretary of the Institute.

New Platinum-Osmium Alloy.—The limited supply and high price of iridium, which is usually used to produce "hard platinum," has led Dr. Fritz Zimmermann, of Newark, N. J., to experiment with other metals for the purpose of producing a hard alloy of platinum. He has found that osmium makes such an alloy, and that much less osmium than iridium is required to produce alloys of similar properties. From ½ per cent to 10 per cent osmium will yield an alloy with platinum which has the hardness, tensile strength and electrical properties obtained when a much larger percentage of iridium is used. The osmium-platinum alloy must be prepared from refined metals. The formula has recently been patented.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in March, 1913, was 60. The total quantity of ore milled during the period was 2,373,273 tons. There were 9,987 stamps in operation with an average duty of 8.8 tons per 24 hours. Tube mills in commission numbered 285. The yield for the month was 790,552 fine ounces of gold. The yield per ton milled was 6.5 dwt. gold.

The Manufacture of Petroleum Products*

By Dr. F. C. Robinson,

Chief Chemist of the Atlantic Refining Co.

It is a real privilege to address you, a body of engineers, on the present subject—the manufacture of petroleum products—because all of these products are or might be used in some engineering operation. I have frequently had occasion to inform my friends that the oil refinery does not manufacture dyes, acetanilid and allied products, nor perfumes. We do not make perfumes.

The first group of products from petroleum is made up of gasolines and naphthas. There is some confusion among the various names, benzine, gasoline, naphtha, etc., but the best practice is to use the word gasoline for any mixture of light hydrocarbons intended for use in any kind of vaporizer, *i. e.*, to be gasified in a gas machine, gasoline torch, gasoline stove or automobile carbureter. Also to confine the word naphtha to mixtures of hydrocarbons intended for some purpose that requires a very good odor, such as the naphtha used by cleaners, varnish makers, soap makers, etc. In this scheme, the word benzine finds no place. Gasolines and naphthas vary in average boiling point according to the use for which they are intended, but all lay between 125 deg. F. and 280 deg. F. In all cases it is essential that they be free from all heavy hydrocarbons that do not evaporate from the hand.

The next group consists of several grades of lamp oil. Lamp oil is a mixture of hydrocarbons whose average boiling point is about 450 deg. F., entirely freed on the one hand from gasoline or naphtha and on the other hand from the heavy hydrocarbons that belong to gas oil and lubricating oil and that would make the oil act badly in the lamp.

The next class is gas oil. While oils of all degrees of volatility have been used, the most economical for the gas maker consists of a mixture of heavy hydro-carbons with an average boiling point of 600 deg. to 650 deg. F. It must be practically free from gasoline and lamp oil on the one hand and from the heavy lubricating oils and asphalt on the other.

The next group is fuel oil. This oil occupies a peculiar position. It must be oil and must not contain gasoline and must be of such a consistency that it can be pumped through pipes and burners, but except for these restrictions one oil is practically as good as another for fuel. The light oils have a slightly higher heat of combustion per pound, but the heavier oils have a slightly higher heat of combustion per gallon. For some purposes, such as oil engines, special oils are required, but, in general, fuel oil is made up of oils that cannot be used for any better purpose.

The next group is that of spindle oils—neutral and paraffin oils. This important group includes hundreds of light lubricating oils designed for use on thousands of different light machines, including gas engines. They must be free from gasoline and lamp oil in order that their flash test shall be high enough to prevent loss by evaporation. The important point to that is that they shall have the proper viscosity for the use intended.

The next group is that of steam cylinder oils, which consist of the heaviest hydrocarbons contained in certain crude oils. In this case also the flash must be such that the oils will not evaporate in a steam cylinder and must have the proper viscosity for the use intended at the temperature of the cylinder.

The next group—paraffin wax—consists of a mixture of hydrocarbons of the paraffin series about $C_{20}H_{42}$ to $C_{35}H_{72}$. The commercial article is rated according to the melting point, which varies from 100 deg. to 135 deg. F.

The next group—vaseline or petrolatum—consists of the higher members of the paraffin series which settle from crude oil mixed and inseparable from some of the oily constituents of the crude. It is marketed as the light-colored material used in medicine and for toilet purposes or as the dark-colored sticky material used in large quantities by the makers of oiled paper.

The next group—the dust laying oils—consists of petroleum asphalt in solution in oils similar to gas oil. The basic idea in their manufacture is that the solvent will slowly evaporate, leaving the dust particles covered with a sticky adherent film. These oils have proven successful as a cheap means of laying dust.

The next group—the road binders—consists of petroleum asphalt properly fluxed with heavy petroleum oils that will not evaporate and of such qualities that they will bind the road materials together both in summer and winter.

The next group—coke—contains but one member. This material being almost entirely free from ash, is used very extensively by makers of electric carbons.

1. These are the desired products. Now let us look at the crude oils from which they are obtained.

2. There are about as many varieties of crude oil as there are oil fields, but the refiner recognizes three distinct types, because each type must be handled by different methods. Viz., (1) The paraffin base crude similar to that found in Pennsylvania and West Virginia and being essentially light colored crudes containing paraffin. (2) Asphalt base crudes similar to those found in Texas and California and being essentially black and containing no paraffin. (3) Mixed base crudes similar to those found from Ohio to Oklahoma and being essentially mixtures of paraffin and asphalt base crudes.

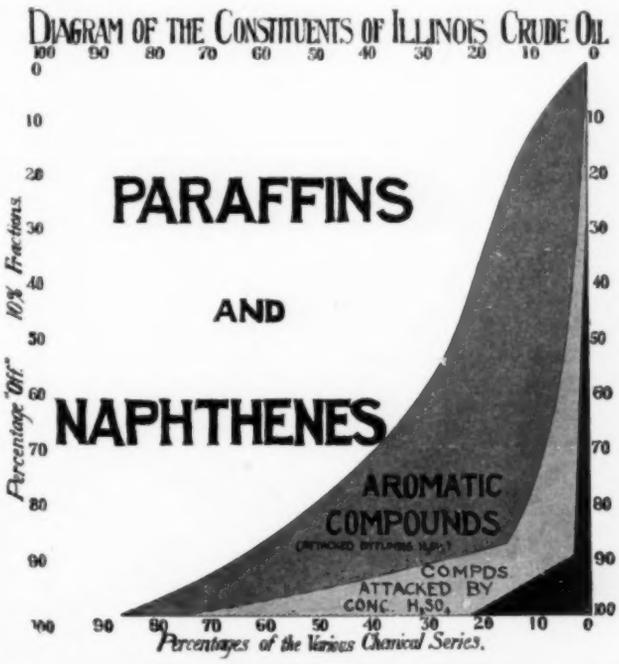


FIG. 1—DIAGRAM OF THE CONSTITUENTS OF ILLINOIS CRUDE OIL

3. In order to obtain some idea of the chemical and physical nature of the crude oil let us imagine a sample of mixed base crude brought into the laboratory for a thorough examination. The chemist would probably distill the sample in a vacuum or in some similar manner in order to avoid destructive distillation and would save the various fractions separate. He would not distill off more than 90 per cent because the heaviest 10 per cent cannot be distilled without breaking it down into simpler molecules. He will then start to examine the various fractions and will keep a record similar to Fig. 1. The horizontal lines indicate the ten fractions.

The first fraction will be a light mobile mixture of hydrocarbons whose average boiling point is about 227 deg. F. The second is a slightly darker and slightly less mobile mixture of hydrocarbons whose average boiling point is about 295 deg. F. The third cut again darker, heavier and less mobile, boiling

*A paper read before the Engineers' Club of Philadelphia on December 21, 1912. From the April, 1913, issue of the *Proceedings* of the Engineers' Club of Philadelphia.

point 369 deg. F. The fourth cut still heavier and 460 deg. F. boiling point. The fifth cut is about 530 deg. F. boiling point. The remaining cuts are increasingly heavier, more viscous and darker in color and the residue in the still is a soft pitch.

The chemist now asks what these ten fractions are chemically and he recognizes four groups of compounds in each fraction that the refiner may have to isolate or remove. First the coloring matter indicated in the diagram by the black area.

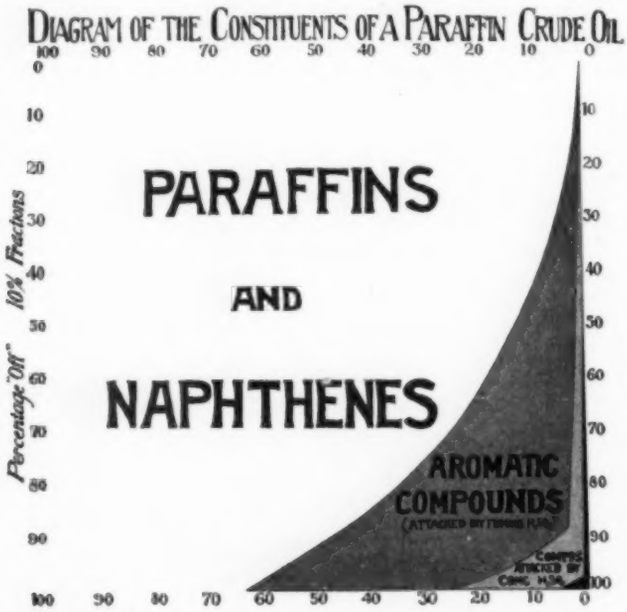


FIG. 2—DIAGRAM OF THE CONSTITUENTS OF A PARAFFIN CRUDE OIL

He can isolate this group by bone black or fuller's earth. When isolated in a pure state it is a jet black brittle compound which is very similar to the purest asphalt.

A second important class of compounds is the material soluble in strong sulphuric acid. The low boiling members of this group represent the odor-bearing compounds of the crude, while the higher members are rich in sulphur, and are easily oxidized. The refiner frequently has to remove a portion of this class.

A third group is that of aromatic hydrocarbons, benzol, naphthalene, anthracene, etc., which may be removed by agitating the oil with fuming sulphuric acid.

The remainder of the crude oil unattacked by fuming sulphuric acid is made up of the naphthene and paraffin series. You will notice that the amount of coloring is very small in the low boiling fractions, but very high in the high boiling fractions, and conversely that the amount of paraffins and naphthenes in the low boiling fractions is very large while in the residue it is very small.

Looking now at similar diagrams for a paraffin base crude (Fig. 2) and an asphalt base crude (Fig. 3); you will notice first that the paraffin base crude contains very little, the asphalt base crude very much color or asphalt. Also that the paraffin base crude contains very much and the asphalt base crude very little paraffin and naphthenes. Also that the paraffin base crude contains very little and the asphalt base crude very much of the aromatic group. Such information as these charts set forth enable the refiner to decide what products he can make and indicate in a general way the methods he must follow. For example the charts would show: (1) The percentage of gasoline that the crude contains; (2) the percentage of lamp oil; (3) the percentage of lubricating oil, etc. However, you will please understand that this is not a practical assay of crude oil. It is merely an attempt to give you a chemical picture, so to speak, showing the distribution of the four important groups that the refiner has to deal with; also that each group consists of thousands of individual hydrocarbons that are never isolated even in the laboratory.

4. Now, starting with crude oil, it is the task of the refiner to isolate the commercial products that I have shown you. You will see the processes outlined on these two charts, and I wish to make especially clear that they represent two distinctly different methods of refining, so much so that one refiner may decide to use one of them to the exclusion of the other, or he may decide to use both of them.

He will be guided in this decision by his local conditions. If it be his desire to produce the maximum amount of gasoline and lamp oil he will use the method marked "Cracking Distillation." (Fig. 4.) If, on the other hand, he wishes to produce the maximum yield of the heavy lubricating oils and petroleum asphalts, he will use the method marked "Fractional Distillation," (Fig. 5), which means that he will simply separate from the crude oil the various fractions which compose it, while the refiner who uses the cracking process actually breaks down these heavy fractions by destructive distillation in a manner similar to the production of benzol and gas by the destructive distillation of coal.

The first step in the cracking process is the cracking distillation. The crude oil is pumped into stills containing 500 to 1000 barrels which consist simply of horizontal steel cylinders made of sheets of half-inch boiler steel riveted together and provided with manholes on top and ends; with pipe for pumping oil into stills; with vapor line for conducting the distilled vapors to a condenser; with safety valve; with combustion chamber underneath; with fractional air condensers and with water condenser, and with pipe for conducting away the gases involved during the distillation.

Such a still is nearly filled with, *e. g.*, mid continent crude oil and fires are lighted. When the temperature of the oil in the still has reached 175 deg. to 200 deg. F. some gases, consisting largely of britane and pentane, are given off and presently the highest naphtha starts to distill over. The firing is continued; the temperature in the still becomes gradually

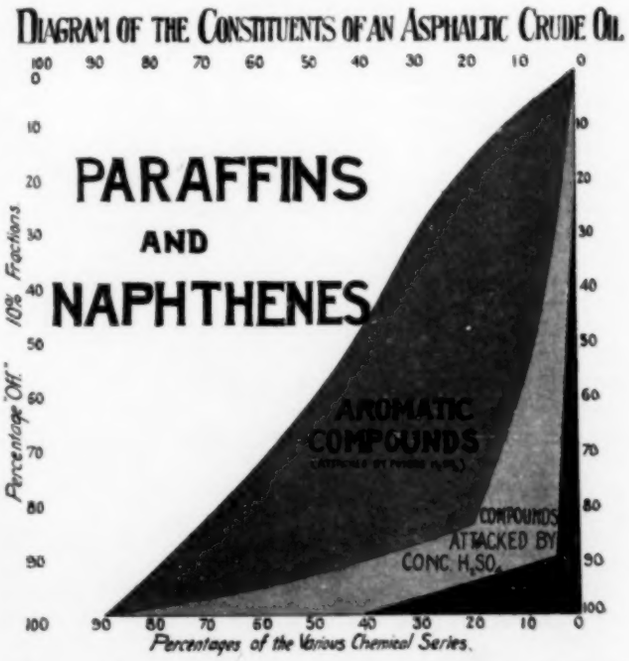


FIG. 3—DIAGRAM OF THE CONSTITUENTS OF AN ASPHALTIC CRUDE OIL

higher; the distillate becomes gradualy heavier until the temperature in the still reaches about 325 deg. F., at which point about 6 per cent or 8 per cent of crude naphtha (200 deg. F. boiling point) has distilled over. This is set aside as crude naphtha.

The distillation is continued until the temperature in the still has reached about 475 deg. F. for crude heavy naphtha and represents 13 per cent to 15 per cent of the crude and has

an average boiling point of about 300 deg. F. The distillation is then continued until the temperature in the still has reached about 625 deg. F. for natural lamp distillate, which represents about 16 per cent to 18 per cent and has an average boiling point of about 450 deg. F.

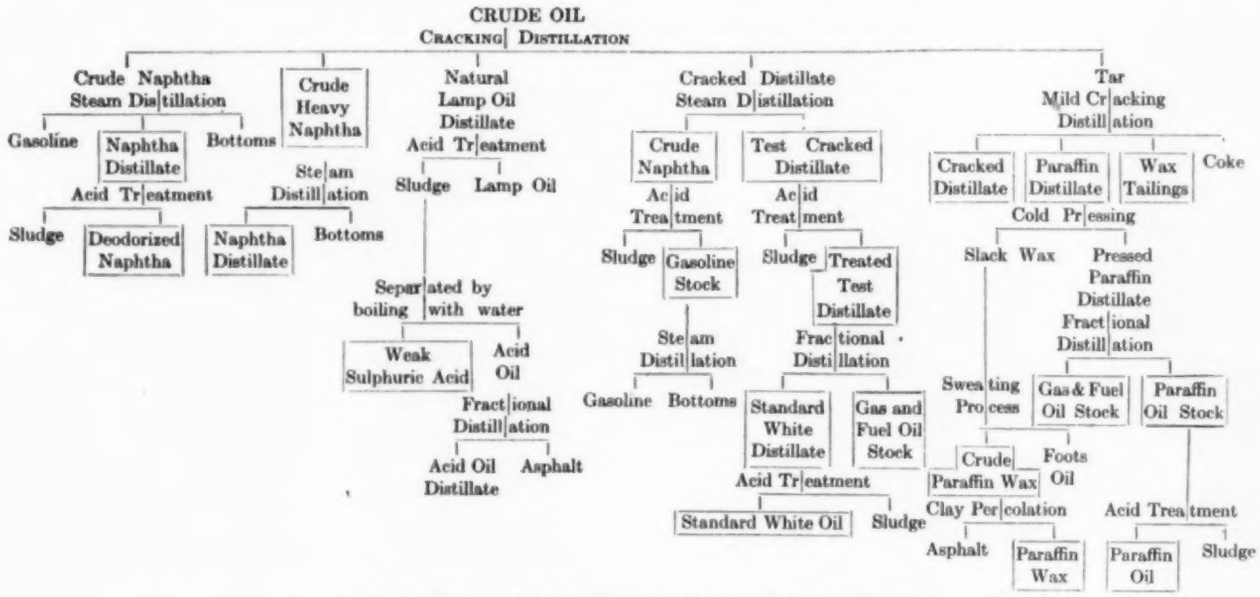


FIG. 4—DIAGRAM OF CRACKING DISTILLATION OF CRUDE OIL

When the still has reached this temperature, cracking or destructive distillation sets in. The fires are slackened in order to distill very slowly and this slow distillation is continued until the temperature in the still reaches 675 deg. to 700 deg. F., producing a distillate with an average boiling point of about 550 deg. F., but containing some gasoline, some lamp oil and much heavier oil which I have designated as gas and fuel oil stock. The yield of this oil is about 20 per cent.

You will notice that this cracking distillation is very different from an ordinary fractional distillation; heavy molecules have been broken down into lighter ones by submitting them to temperatures at which they are unstable.

There yet remains in the still a heavy black tar representing about 42 per cent of the crude oil. This is the source of paraffin wax and the line of lubricating oils called paraffin oils. It is no longer desirable to carry on a cracking distillation because this would result in the destruction of the valuable products desired. The distillation is continued in such a way as to avoid cracking as much as possible (is distilled fast) either in the same still or more commonly in separate smaller stills called tar stills.

This tar still distillation is carried on very rapidly in order to produce the maximum yield of paraffin distillate (about 22 per cent). In addition to the paraffin distillate, there is also produced by destructive distillation about 15 per cent of cracked distillate. At the end of the distillation the stream becomes so heavy that it will sink in water and is then known as wax tailings, which amounts to about 1 per cent of the crude oil. When the distillation stops there remains in the still nothing but coke, amounting to about 4 per cent of the crude oil.

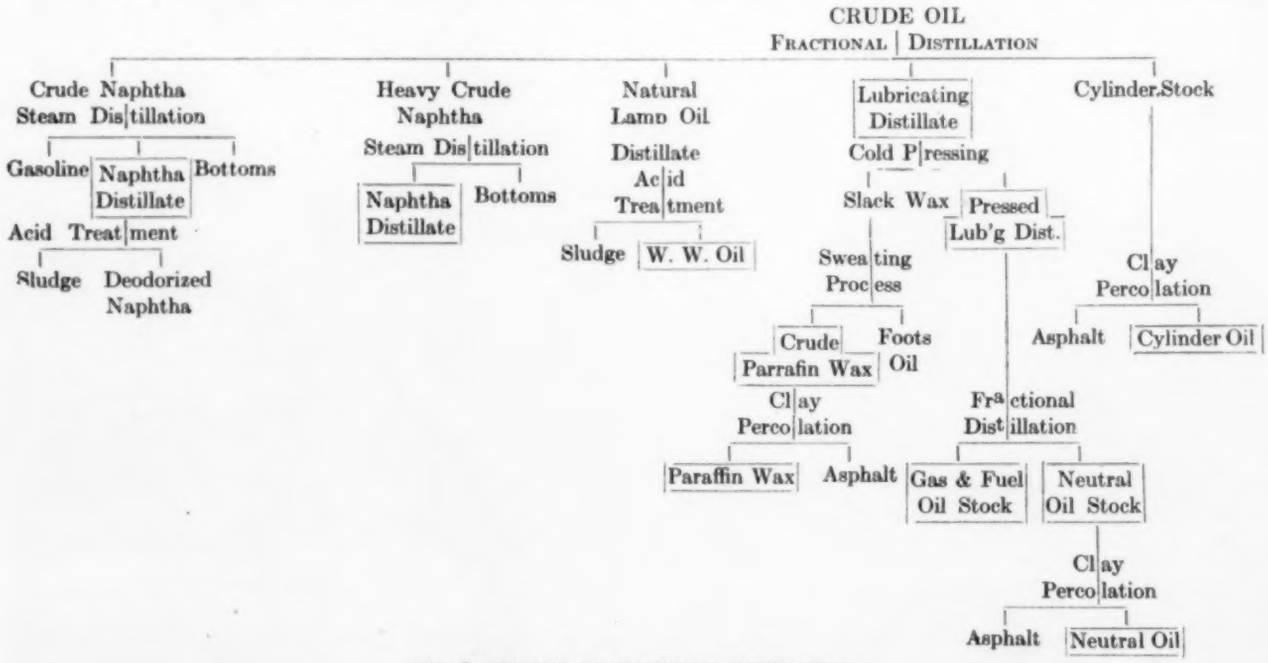


FIG. 5—DIAGRAM OF FRACTIONAL DISTILLATION

Now taking up the various fractions; first the crude naphtha. This is again distilled; first, in order to separate it into

the various gasolines and naphthas that compose it and secondly to separate it from the small amount of bottoms or light lamp oil that it contains. This is done in a still which is heated by steam usually by injecting live steam directly into the gasoline.

When the distillation starts, some gas is given off, then the lightest distillate appears at the trap usually about 90 deg.

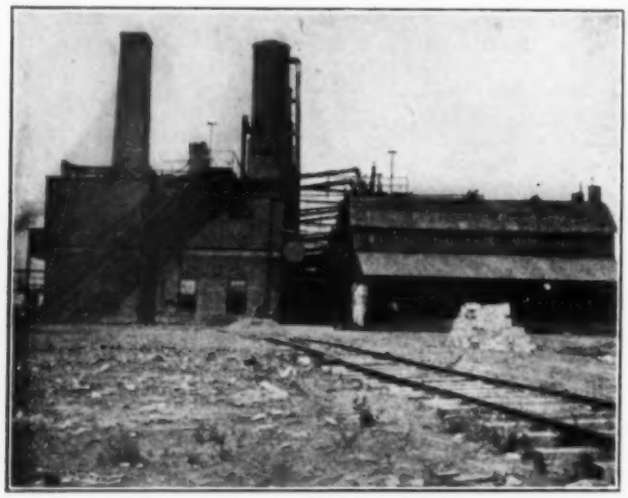


FIG. 6—CRUDE STILL

Beaume gravity. The distillate gradually gets heavier until all the gasoline has distilled off. The receiver is then changed and the naphtha distillate is separated. At this point about 90 per cent has distilled off leaving a bottom about 10 per cent. This bottom is essentially lamp oil and is used as such. The heavy crude naphtha is handled in the same manner except that it contains little or no gasoline and contains about 50 per cent of bottom or lamp oil. The cracked distillate is also distilled with steam to remove about 4 per cent of crude naphtha.

Up to this point all the operations have been different types of distillation. The next step in handling the naphtha distillate from both sources, the lamp oil distillate and the crude naphtha from cracked distillate and the test cracked distillate, is the acid treatment. You have seen in the crude diagrams that all the fractions contain a certain percentage of material attackable by sulphuric acid so that this reagent affords a convenient means for removing color and odor from the remainder of the hydrocarbon distillate.

In practice the naphtha distillates are agitated with about 5 per cent by volume, the lamp oil distillates with about 1.5 per cent of sulphuric acid, oil of vitriol, for about a half hour. The color and odor bearing compounds combine with the acid producing a heavy black viscous mass called acid sludge, which settles to the bottom of the vessel. The sludge is drawn off and the oil washed with water and alkali to remove all traces of acid and is then ready for the market.

The sludge from all acid treatments is separated into unstable products. This is accomplished by boiling it with water, which results in the dilution of the acid and renders it incapable of holding in solution the impurities.

The weak acid (30 deg. to 50 deg. Beaume) settles to the bottom, is drawn off and reconcentrated. The upper layer consisting of the impurities is known as acid oil. This acid oil is separated by fractional distillation into a light distillate which consists of all the evil odors that the original distillate contained and a residue consisting of the asphaltic compounds that were removed by the acid.

You now have before you all the processes used in the manufacture of naphtha and lamp oil.

The next general subject is the handling of the paraffin distillate, which is the direct source of the paraffin wax and all of the paraffin oils. The first step is the process of cold pressing. The distillate is first cooled to 20 deg. to 30 deg. F. by pumping through pipes surrounded by cold brine, thereby causing the paraffin wax (amounting to about 10 per cent of the distillate) to solidify. This solid 10 per cent mixed with the 90 per cent liquid oil, forms a soft mush which is pumped through a filter press. That which stays in the press is called the slack wax and amounts to about 20 per cent of the paraffin distillate. The 80 per cent that goes through is called pressed distillate.

The slack wax, consisting of about equal parts of oil and wax, is then put through a process peculiar to the oil business, known as the sweating process. It consists of cooling the mixture until it has become a solid cake and then very gradually warming it. The crystals of the paraffi form a network through which the oil is distributed and when the mass is warmed the oil sweats out and drips away. It always carries with it some wax in solution, but the final result is that the oil all sweats out leaving the paraffin wax in a fairly pure state.

This sweating process separates the slack wax into crude paraffin wax and what is known as Foots oil. The latter still contains much paraffin, which is removed by putting it again through either the cold pressing or sweating process.

The crude paraffin wax is then put through another process that is peculiar to the oil business, that of clay or bone black percolation for the purpose of removing asphaltic coloring matter and thereby changing the crude paraffin to refined colorless paraffin. The clay used for this purpose has properties similar to those of bone black, *i. e.*, it absorbs and retains tarry and asphaltic compounds. It is found in Florida and Georgia where it is mined, roasted, broken up and sifted to separate grains such as you see here. It is very porous and light, weighing only about 2.3 as much as water. This clay, or fuller's earth, as it is commonly called, is put into large upright cylinders holding 10 or 20 tons and provided with a finely perforated bottom. The crude wax is melted and poured on top of the clay. It trickles down through the clay bed and passes through the perforated bottom.

The first drippings from such a filter bed are absolutely colorless, but as the filtration progresses, the color becomes more and more like crude wax. A ton of clay yields 5 or 6 tons of first-quality paraffin wax. The amount of asphalt or coloring

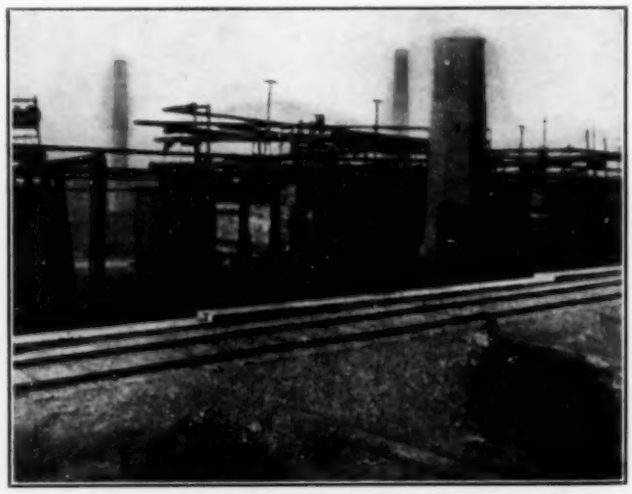


FIG. 7—TAR STILL

matter retained by the clay is exceedingly small and is removed by burning the clay in a cement kiln of the usual type, thus regenerating the clay for subsequent use.

I have explained that the paraffin distillate is the source of paraffin wax and light lubricating oils and have explained the various steps in the preparation of paraffin. The light lubricating oils are made from the filtrate from the cold presses—the pressed paraffin distillate—by putting it through

the process of fractional distillation, thereby separating it into a distillate of light oils that go to make up the gas and fuel oil stock and a residue in the still called paraffin oil stock, representing from 15 per cent to 50 per cent of the charge of the still depending on the quality desired.

This type of distillation is also peculiar to the oil business. The desired product is a heavy oil so that all cracking must be avoided in order to produce the maximum yield of this oil.

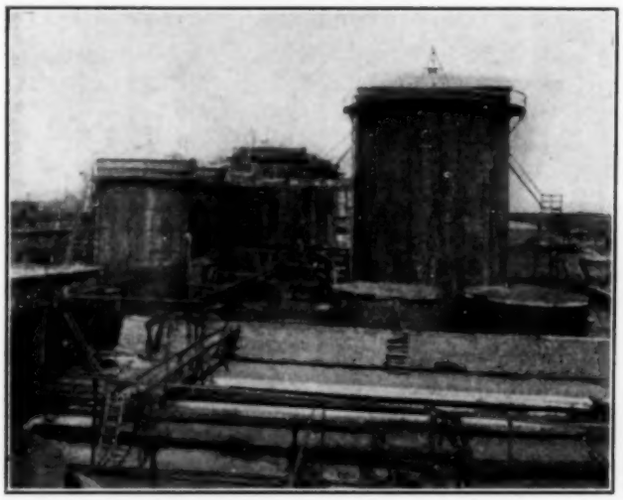


FIG. 8—AGITATORS

The result is accomplished by using the very important process of distillation with bottom steam or fractional distillation. A still is charged with the pressed paraffin distillate and fires are lighted. The temperature in the still rises and when it has reached about 400 deg. F., the distillation begins.

Shortly after this the live steam is injected into the oil in distillation through perforated pipes placed near the bottom of the still. No water accumulates in the still because the temperature is so high. The steam passes upward through the oil as an inert gas and passes through the condenser with the oil vapors and is condensed there with the oil. The effect of this current of steam through the oil is exactly that of a vacuum distillation, *i. e.*, it lowers the boiling point of the oil in distillation and allows a heavy oil to be distilled at temperatures below the temperature of destructive distillation. You will notice that cracking sets in at about 630 deg. F. Without the use of steam, the distillation in question would require that the still be heated to about 750 deg. F. This would result in the destruction of the desired oil. With steam it can be carried on with a maximum temperature of 600 deg. F., thus entirely avoiding destructive distillation.

The paraffin oil stock is a dark colored unattractive looking material which is transformed into the valuable paraffin oils of commerce by treating with sulphuric acid in the manner already described. The treating loss in this case is from 10 per cent to 30 per cent. The whole cracking process described thus far is designed to produce the maximum yield of gasoline and lamp oil from crudes containing asphalt.

We will now turn to the method of refining the light colored non-asphaltic crude oils from which the valuable cylinder oils may be made. The object in this case is to avoid all destructive distillation in order to produce the maximum yield of the very heavy lubricating oils. The still is charged with the crude oil, fires are lighted, the crude naphtha is distilled off as in the other distillations, but when the temperature is well above the boiling point of water, steam is injected into the oil as before described.

Under these conditions the crude naphtha has distilled off when the temperature in the still has reached about 280 deg. F., while without steam the still temperature was about 375 deg. F. The yield from this kind of crude is about 13 per cent.

The heating is continued, more and more steam being injected, the distillate becoming heavier and heavier until the heavy crude naphtha has distilled off. At this point the temperature in the still has reached about 330 deg. F., while without steam at this point the temperature was 475 deg. F. The yield of this oil is about 13 per cent.

The distillation is still continued until the natural lamp oil distillate has distillated off. At this point the distillate in the still is only 500 deg. F., while without steam it was 630 deg. F.

Now instead of trying to increase cracking by slowing the rate of distillation, the amount of steam is increased and the distillation is carried on as fast as possible to avoid cracking. This is continued until the lubricating distillate, amounting to about 28 per cent of the crude has distilled over. At this point the temperature in the still has reached about 620 deg. F., and the distillation is stopped, leaving the cylinder oil stock in the still.

You will recall that in the cracking process the temperature in the still reached 850 deg. F., and yielded a residue in the still of solid coke. This whole distillation has been conducted in such a manner as to produce the best quality and largest yield of this residual oil, the source of all steam cylinder oils.

The various fractions from the stills are put through exactly the same processes as the corresponding fractions from the cracking distillation, except the lubricating distillate. The crude naphtha and the crude heavy naphtha are fractioned and some of the distillates treated with acid.

The lamp oil distillate is treated with acid as before described. The lubricating distillate, amounting to about 28 per cent of the crude, is put through the same process for the removal of wax and gas oil that the paraffin oil was put through, the only difference being that the resulting lubricating stocks are not usually treated with sulphuric acid, but are brought to the desired color by percolating through fuller's earth to produce the neutral spindle oils of commerce.

The cylinder stock remaining in the still, and amounting to 8 per cent to 13 per cent of the crude oil, is ready for shipment as unfiltered cylinder oil. Or it may be percolated through clay to produce the filtered oils of commerce. In order to show the wonderful effects of fuller's earth on the color of oils, I have selected a series of samples showing (1) cylinder stock that was pumped into the filter, (2) the first oil that

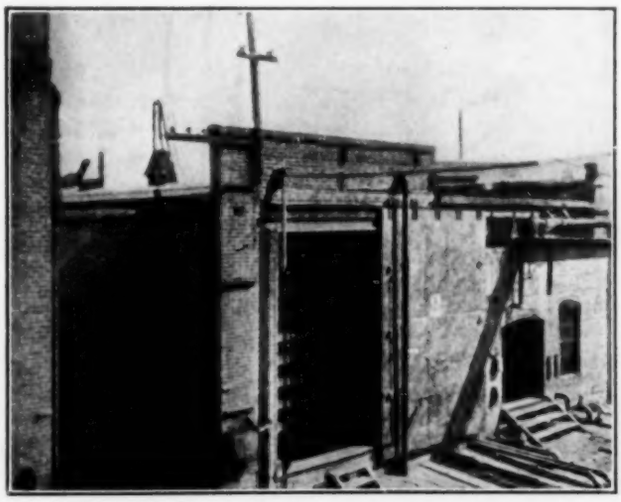


FIG. 9—SWEATER

percolated through, (3) a sample when the percolation was half completed, and (4) a sample at the end of the percolation. At this point the clay must be restored.

You will notice that I have described the cracking distillation of a mixed base crude and the fractional distillation of a paraffin base crude. Now, what would result if the mixed base crude had been put through the process of fractional distillation? The crude naphtha would, of course, distill off, then the

crude heavy naphtha, then the natural lamp oil, then a distillate containing wax and lubricating oil corresponding to the lubricating distillate, leaving in the still a residue consisting of a soft pitch and amounting to about 15 per cent of the crude oil. The maximum temperature in the still would be about 630 deg. F. This residue has the property of being vulcanized by the action of air at about 500 deg. F., or by sulphur at about 400 deg. to 450 deg. F. This process is carried on in a still and consists in heating the residue to the desired temperature and pumping air through the mass for one to two days, depending on the hardness desired. The resulting hard pitch is the petroleum road binder of commerce.

* * *

In the discussion which followed Mr. Price asked as to the probable future supply of fuel oil or gasoline. Mr. Irish said that "the supply of fuel and gas oils must necessarily be a function of the crude oil production of the world. While the amount of crude oil produced has within the past few years been greater than at any other time since the discovery of petroleum, and consequently indicates an ever increasing supply, we are at present and have been for the last two or three years, suffering from a falling off in our production. We are informed that the crude oil stocks in the United States, east of the Rocky Mountains, are decreasing at the rate of about 40,000 barrels a day.

"That means to us that the supply has fallen off and the demand has increased, until the demand has overtaken the production, and we are drawing on accumulated stocks, and so long as that continues, exhaustion of the supply of gas and fuel oil will, of course, come nearer.

"But the supply of those oils has been supplemented by the Mexican crude. . . .

"The Mexican crude is new to the refiner, and it would be unsafe and unwise to predict what can be made from it. We know it has an asphalt base, and is limited in possibilities to the production of those things Mr. Robinson has described as being peculiar to the asphalt base crude oil. It means that a large proportion of such crude oil will find its way into the fuel oil market as raipdly as it can be brought into this country."

In reply to a question by Mr. J. C. Parker as to the length of pipe lines Mr. Irish replied that the pipe line system is complete from Oklahoma to the Atlantic coast, so that Oklahoma crudes are distributed in New York and Philadelphia.

Temperature Conversion Table

In our March issue, 1910 (Vol. VIII, p. 123) we published a temperature conversion table by Dr. **Leonard Waldo** for converting Centigrade into Fahrenheit degrees.

In the *Bulletin of the American Institute of Mining Engineers*, of April, 1913, Dr. Waldo publishes a second table for conversion from Fahrenheit to Centigrade degrees. This is herewith reproduced. Like his former table it is based on the arrangement well known from logarithmic tables. Besides convenience it has the advantage of exactness.

As the table will be found almost self-explanatory, the following examples will be sufficient to explain its use:

$-246.0° \text{ F.} = -151.11° \text{ C.} - 3.33° \text{ C.} = -154.44° \text{ C.}$

$3762° \text{ F.} = 2071.11° \text{ C.} + 1.11° \text{ C.} = 2072.22° \text{ C.}$

$2423.5° \text{ F.} = 1326.666° \text{ C.} + 1.666° \text{ C.} + 0.277° \text{ C.} = 1328.609° \text{ C.}$

In his paper in the *Bulletin* of the American Institute of Mining Engineers Dr. Waldo remarks that for mental calculation the easiest rule to convert Centigrade into Fahrenheit is to add 32° to twice the Centigrade reading less one-tenth of the product:

Thus $1000° \text{ C.} = 2000° - 200° + 32° = 1832° \text{ F.}$, and to convert Fahrenheit to Centigrade, subtract 32° from the Fahrenheit reading, multiply by 10 and divide by 2 and 9 successively:

Thus $1000° \text{ F.} = (1000 - 32) \times 10 \div 2 \div 9 = 537.77° \text{ C.}$

Another and often very convenient method is by use of a graphic construction. A diagram illustrating this graphical method is given in the paper.

Dots over figures indicate recurring decimals.

F.°	0	10	20	30	40	50	60	70	80	90
	C.	C.	C.	C.	C.	C.	C.	C.	C.	C.
−250	−240.0	−245.5	−251.1	−256.6	−262.2	−267.7	.....	.....	.....	.....
−200	−184.4	−190.0	−195.5	−201.1	−206.6	−212.2	−217.7	−223.3	−228.8	−234.4
−150	−128.8	−134.4	−140.0	−145.5	−151.1	−156.6	−162.2	−167.7	−173.3	−178.8
−100	−73.3	−78.8	−84.4	−90.0	−95.5	−101.1	−106.6	−112.2	−117.7	−123.3
−0	−17.7	−23.3	−28.8	−34.4	−40.0	−45.5	−51.1	−56.6	−62.2	−67.7
0	−17.7	−12.2	−6.6	−1.1	+4.4	+10.0	+15.5	+21.1	+26.6	+32.2
100	37.7	43.3	48.8	54.4	60.0	65.5	71.1	76.6	82.2	87.7
200	93.3	98.8	104.4	110.0	115.5	121.1	126.6	132.2	137.7	143.3
300	148.8	154.4	160.0	165.5	171.1	176.6	182.2	187.7	193.3	198.8
400	204.4	210.0	215.5	221.1	226.6	232.2	237.7	243.3	248.8	254.4
500	260.0	265.5	271.1	276.6	282.2	287.7	293.3	298.8	304.4	310.0
600	315.5	321.1	326.6	332.2	337.7	343.3	348.8	354.4	360.0	365.5
700	371.1	376.6	382.2	387.7	393.3	398.8	404.4	410.0	415.5	421.1
800	426.6	432.2	437.7	443.3	448.8	454.4	460.0	465.5	471.1	476.6
900	482.2	487.7	493.3	498.8	504.4	510.0	515.5	521.1	526.6	532.2
1000	537.7	543.3	548.8	554.4	560.0	565.5	571.1	576.6	582.2	587.7
1100	593.3	598.8	604.4	610.0	615.5	621.1	626.6	632.2	637.7	643.3
1200	648.8	654.4	660.0	665.5	671.1	676.6	682.2	687.7	693.3	698.8
1300	704.4	710.0	715.5	721.1	726.6	732.2	737.7	743.3	748.8	754.4
1400	760.0	765.5	771.1	776.6	782.2	787.7	793.3	798.8	804.4	810.0
1500	815.5	821.1	826.6	832.2	837.7	843.3	848.8	854.4	860.0	865.5
1600	871.1	876.6	882.2	887.7	893.3	898.8	904.4	910.0	915.5	921.1
1700	926.6	932.2	937.7	943.3	948.8	954.4	960.0	965.5	971.1	976.6
1800	982.2	987.7	993.3	998.8	1004.4	1010.0	1015.5	1021.1	1026.6	1032.2
1900	1037.7	1043.3	1048.8	1054.4	1060.0	1065.5	1071.1	1076.6	1082.2	1087.7
2000	1093.3	1098.8	1104.4	1110.0	1115.5	1121.1	1126.6	1132.2	1137.7	1143.3
2100	1148.8	1154.4	1160.0	1165.5	1171.1	1176.6	1182.2	1187.7	1193.3	1198.8
2200	1204.4	1210.0	1215.5	1221.1	1226.6	1232.2	1237.7	1243.3	1248.8	1254.4
2300	1260.0	1265.5	1271.1	1276.6	1282.2	1287.7	1293.3	1298.8	1304.4	1310.0
2400	1315.5	1321.1	1326.6	1332.2	1337.7	1343.3	1348.8	1354.4	1360.0	1365.5
2500	1371.1	1376.6	1382.2	1387.7	1393.3	1398.8	1404.4	1410.0	1415.5	1421.1
2600	1426.6	1432.2	1437.7	1443.3	1448.8	1454.4	1460.0	1465.5	1471.1	1476.6
2700	1482.2	1487.7	1493.3	1498.8	1504.4	1510.0	1515.5	1521.1	1526.6	1532.2
2800	1537.7	1543.3	1548.8	1554.4	1560.0	1565.5	1571.1	1576.6	1582.2	1587.7
2900	1593.3	1598.8	1604.4	1610.0	1615.5	1621.1	1626.6	1632.2	1637.7	1643.3
3000	1648.8	1654.4	1660.0	1665.5	1671.1	1676.6	1682.2	1687.7	1693.3	1698.8
3100	1704.4	1710.0	1715.5	1721.1	1726.6	1732.2	1737.7	1743.3	1748.8	1754.4
3200	1760.0	1765.5	1771.1	1776.6	1782.2	1787.7	1793.3	1798.8	1804.4	1810.0
3300	1815.5	1821.1	1826.6	1832.2	1837.7	1843.3	1848.8	1854.4	1860.0	1865.5
3400	1871.1	1876.6	1882.2	1887.7	1893.3	1898.8	1904.4	1910.0	1915.5	1921.1
3500	1926.6	1932.2	1937.7	1943.3	1948.8	1954.4	1960.0	1965.5	1971.1	1976.6
3600	1982.2	1987.7	1993.3	1998.8	2004.4	2010.0	2015.5	2021.1	2026.6	2032.2
3700	2037.7	2043.3	2048.8	2054.4	2060.0	2065.5	2071.1	2076.6	2082.2	2087.7
3800	2093.3	2098.8	2104.4	2110.0	2115.5	2121.1	2126.6	2132.2	2137.7	2143.3
3900	2148.8	2154.4	2160.0	2165.5	2171.1	2176.6	2182.2	2187.7	2193.3	2198.8
4000	2204.4	2210.0	2215.5	2221.1	2226.6	2232.2	2237.7	2243.3	2248.8	2254.4
4100	2260.0	2265.5	2271.1	2276.6	2282.2	2287.7	2293.3	2298.8	2304.4	2310.0
4200	2315.5	2321.1	2326.6	2332.2	2337.7	2343.3	2348.8	2354.4	2360.0	2365.5
4300	2371.1	2376.6	2382.2	2387.7	2393.3	2398.8	2404.4	2410.0	2415.5	2421.1
4400	2426.6	2432.2	2437.7	2443.3	2448.8	2454.4	2460.0	2465.5	2471.1	2476.6
4500	2482.2	2487.7	2493.3	2498.8	2504.4	2510.0	2515.5	2521.1	2526.6	2532.2
4600	2537.7	2543.3	2548.8	2554.4	2560.0	2565.5	2571.1	2576.6	2582.2	2587.7
4700	2593.3	2598.8	2604.4	2610.0	2615.5	2621.1	2626.6	2632.2	2637.7	2643.3
4800	2648.8	2654.4	2660.0	2665.5	2671.1	2676.6	2682.2	2687.7	2693.3	2698.8
4900	2704.4	2710.0	2715.5	2721.1	2726.6	2732.2	2737.7	2743.3	2748.8	2754.4
5000	2760.0	2765.5	2771.1	2776.6	2782.2	2787.7	2793.3	2798.8	2804.4	2810.0
5100	2815.5	2821.1	2826.6	2832.2	2837.7	2843.3	2848.8	2854.4	2860.0	2865.5
5200	2871.1	2876.6	2882.2	2887.7	2893.3	2898.8	2904.4	2910.0	2915.5	2921.1
5300	2926.6	2932.2	2937.7	2943.3	2948.8	2954.4	2960.0	2965.5	2971.1	2976.6
5400	2982.2	2987.7	2993.3	2998.8	3004.4	3010.0	3015.5	3021.1	3026.6	3032.2
5500	3037.7	3043.3	3048.8	3054.4	3060.0	3065.5	3071.1	3076.6	3082.2	3087.7
5600	3093.3	3098.8	3104.4	3110.0	3115.5	3121.1	3126.6	3132.2	3137.7	3143.3
5700	3148.8	3154.4	3160.0	3165.5	3171.1	3176.6	3182.2	3187.7	3193.3	3198.8
5800	3204.4	3210.0	3215.5	3221.1	3226.6	3232.2	3237.7	3243.3	3248.8	3254.4
5900	3260.0	3265.5	3271.1	3276.6	3282.2	3287.7	3293.3	3298.8	3304.4	3310.0
6000	3315.5	3321.1	3326.6	3332.2	3337.7	3343.3	3348.8	3354.4	3360.0	3365.5
6100	3371.1	3376.6	3382.2	3387.7	3393.3	3398.8	3404.4	3410.0	3415.5	3421.1
6200	3426.6	3432.2	3437.7	3443.3	3448.8	3454.4	3460.0	3465.5	3471.1	3476.6
6300	3482.2	3487.7	3493.3	3498.8	3504.4	3510.0	3515.5	3521.1	3526.6	3532.2
6400	3537.7	3543.3	3548.8	3554.4	3560.0	3565.5	3571.1	3576.6	3582.2	3587.7
6500	3593.3	3598.8	3604.4	3610.0	3615.5	3621.1	3626.6	3632.2	3637.7	3643.3
6600	3648.8	3654.4	3660.0	3665.5	3671.1	3676.6	3682.2	3687.7	3693.3	3698.8
6700	3704.4	3710.0	3715.5	3721.1	3726.6	3732.2	3737.7	3743.3	3748.8	3754.4
6800	3760.0	3765.5	3771.1	3776.6	3782.2	3787.7	3793.3	3798.8	3804.4	3810.0
6900	3815.5	3821.1	3826.6	3832.2	3837.7	3843.3	3848.8	3854.4	3860.0	3865.5
7000	3871.1	3876.6	3882.2	3887.7	3893.3	3898.8	3904.4	3910.0	3915.5	3921.1
7100	3926.6	3932.2	3937.7	3943.3	3948.8	3954.0	3960.0	3965.5	3971.1	3976.6
7200	3982.2	3987.7	3993.3	3998.8	4004.4	4010.0	4015.5	4021.1	4026.6	4032.2
7300	4037.7	4043.3	4048.8	4054.4	4060.0	4065.5	4071.1	4076.6	4082.2	4087.7
7400	4093.3	4098.8	4104.4	4110.0	4115.5	4121.1	4126.6	4132.2	4137.7	4143.3
7500	4148.8	4154.4	4160.0	4165.5	4171.1	4176.6	4182.2	4187.7	4193.3	4198.8
7600	4204.4	4210.0	4215.5	4221.1	4226.6	4232.2	4237.7	4243.3	4248.8	4254.4
7700	4260.0	4265.5	4271.1	4276.6	4282.2	4287.7	4293.3	4298.8	4304.4	4310.0
7800	4315.5	4321.1	4326.6	4332.2	4337.7	4343.3	4348.8	4354.4	4360.0	4365.5
7900	4371.1	4376.6	4382.2	4387.7	4393.3	4398.8	4404.4	4410.0	4415.5	4421.1
F.°	0	10	20	30	40	50	60	70	80	90

F.°	C.°
1	0.5
2	1.1
3	1.6
4	2.2
5	2.7
6	3.3
7	3.8
8	4.4
9	5.0

F.°	C.°
1	0.5
2	1.1
3	1.6
4	2.2
5	2.7
6	3.3
7	3.8
8	4.4
9	5.0

Cyanide Practice in the Black Hills, South Dakota—I

By H. C. Parmelee.

The small area covered by the Black Hills in southwestern South Dakota holds an important place in the annals of metallurgy. Its development has followed the lines so common to many of the older mining districts, beginning with the simple processes of placering for free gold in the stream beds, continuing through the list of amalgamation, smelting and chlorination, and finally gaining prominence by the successful and extensive use of cyanidation which is today the prevalent method of gold recovery. As in some other western mining districts, also, the high-grade ores which originally attracted the miner have been exhausted, and the stability of the district now rests on the possibility of treating at a profit the large tonnages of more refractory, low-grade ores which remain. Metallurgical success in the Black Hills today depends on good business management combined with technical skill. The best known example is the Homestake Mining Company, the success of which has been a stimulus for many smaller but successful ventures, as well as the basis for some flagrant, illegitimate promotions that have done immeasurable harm. In these notes the practice at the Homestake mills will not be touched upon, as the subject has recently been thoroughly covered by Messrs. Clark and Sharwood,[1] metallurgist and chemist, respectively, for the company. The gist of their article has already appeared in this journal.[2]

General Conditions.

The majority of the ores now treated in the Black Hills range in value from $2 to $10 per ton. Some higher grade ores are still mined, but only in small quantities and incidentally with those of lower grade. In most cases constant assaying of daily mine samples must be resorted to in order to determine whether the ore may be treated at a profit, for it is not possible to be guided wholly by the appearance of the ore, nor to assume that all parts of the same deposit are uniformly valuable. Silver occurs in practically all the ores; sometimes in greater quantity, but usually in less value, than gold.

For a detailed study of the different kinds of ore, the reader is referred to Irving.[3] The ores now being treated are all highly silicious, containing from 70 per cent to 85 per cent silica. They vary in hardness from the "hard blue" ore, which is dense, unoxidized and fine-grained, down through the coarse and porous quartzites, to the soft, oxidized shales. The unoxidized ores contain from 2 per cent to 10 per cent sulphur in the form of pyrite, but the oxidized ores are free from this element and have the characteristic red color of iron oxide. Tungsten in the form of wolframite has been found in commercial quantities, some high-grade gold-tungsten ore having been shipped from the Wasp No. 2. Traces of tungsten throughout some of the quartzites are not uncommon. Stibnite also has been identified, while traces of copper are general. Tellurium occurs, presumably as sylvanite, but the telluride ores are not common, nor do they offer much prospect of becoming an important source of gold.

Power and Labor.

The use of electric power is general throughout the district, which is traversed by the lines of the Consolidated Light & Power Company. The tariff is based on a flat rate of $1 per month per installed horsepower, plus two cents per kw-hour for current consumed. Monthly bills up to $300 are net; 10 per cent discount is allowed on the second $300; 20 per cent on the third $300; and 30 per cent on all over $900. The Homestake company has its own hydro-electric power plant.

Mill labor is well paid, the daily wage ranging from $3 to $3.50 according to the class of labor performed. In some cases the standard wage is increased for specially reliable and efficient service, or for the performance of additional service. Foremen receive from $4 to $4.50.

Grinding in Solution at Golden Reward Mill.

Current practice at the Golden Reward mill in Deadwood may not be taken as fairly representative of the company's ideas and intentions, although the mill equipment includes many of the modern appliances used in cyanidation. An entirely new mill at another location has been contemplated for some time, and many experiments have been and are still being made with a view to adopting the best machinery and methods in the new plant.

The problem at the Golden Reward is similar to that of a custom mill, for the company treats a variety of ores from several of its mines which are under lease. The result is an ever-varying grade, character and quantity of mill feed, in which each class of ore might yield better results if given the special treatment to which it is best suited. The present sand and slime departments cannot, therefore, give the same results which would obtain on a uniform feed. Some of the ore is soft and oxidized, forming a large proportion of slime; another class is hard, dense and refractory in extraction; and still another is clean quartzite, forming little slime, and easily leached in the sand tanks. The grade varies from $4 to $15 per ton, and probably averages $7 to $8.

Ore Weighed and Sampled.

As delivered to the mill, the ore is weighed in carload lots, crushed dry by gyratory and rolls, and automatically sampled by buckets attached to a chain traveling through the falling stream of ore. This system gives an accurate sample of the mill feed, provided the stream of ore is falling uniformly and continuously.

Lime is added dry at the crusher, where the ore is reduced to about 1 in. size and elevated to storage bins above the Chilean mills. The latter receive a continuous feed by means of a modified Challenge feeder. Mill solution is added and the ore is ground to pass ton-cap screen having slots 0.021 in. wide. Under these conditions the Chilean mills show a capacity of 4.5 to 5 tons per hour. The discharged pulp is then classified into sand and slime in Akins and drag classifiers. The latter were used altogether prior to the introduction of the Akins, but gave unsatisfactory results under the heavy load imposed on them. At present only one remains and is doing fair work under a light load.

Screen Analyses on Chilean and Classifier Products.

The conditions for the following tests were: Chilean feed, 1 in. and finer; mill speed, 30 r.p.m.; screen, ton-cap 0.026-in. slot; capacity, 5.2 tons ore per hour; about 6 tons solution per ton of ore; one-third of Chilean discharge diverted to drag classifier and two-thirds to Akins, this being the condition under which the drag operated at highest efficiency. Even under these conditions it will be apparent from the figures below that the drag classifier is not as efficient as the more modern machine.

Owing to the variable physical character of the ore it is difficult to give consistent figures on screen life and metal consumption in the Chilean mills. Ton-cap screen has been found best suited to local conditions. Three different makes of steel have been used for rolls and dies, viz., Midvale, Standard and Chrome. The latter has shown varying life in different sets, but is used as being less expensive. A set of three rolls and one die of the different brands have crushed the following tonnages of ore before removal due to wear: Midvale, 9049 tons; Standard, 9347 tons; Chrome, (three sets) low, 6435; high, 9521; average, 8083 tons.

Sand Treatment.

The classified sand is distributed in the ordinary way to leaching tanks. The cycle of treatment varies according to the proportions of sand and slime being made, and it is customary to treat the sand as long as the tanks are not required for new charges. The first solution applied is that decanted

[1] *Bulletin* No. 98, Inst. Min. and Met.
[2] January, 1913, pp. 50-51; February, 1913, pp. 105-106.
[3] U. S. G. S. *Bulletin* No. 225, pp. 123.

CHILEAN MILL DISCHARGE

Mesh	Per Cent
+ 20	0.155
+ 40	6.78
+ 60	9.195
+ 80	3.28
+100	6.89
+200	20.08
−200	53.62
	100.00

CLASSIFIER SAND PRODUCTS

Mesh	Drag %	Akins %
+ 20	1.30	1.33
+ 40	23.60	25.16
+ 60	17.94	25.98
+ 80	3.81	3.01
+100	13.81	10.43
+200	28.63	23.78
−200	10.91	10.31
	100.00	100.00

CLASSIFIER SLIME PRODUCTS

Mesh	Drag %	Akins %
+ 20	0.00	0.00
+ 40	1.58	0.23
+ 60	1.16	0.44
+ 80	0.49	0.27
+100	0.78	0.65
+200	3.90	7.60
−200	92.09	90.81
	100.00	100.00

from the thickening slime. This is an unusual procedure, for one would expect to find this solution flowing directly to the precipitation department rather than used as a leaching solution. The sand tanks, however, act as clarifiers of the solution, and the value of the latter is materially enhanced in its passage through the sand. This first application continues for three days, after which successive washes are applied, bringing the total time of sand treatment up to 14 or possibly 15 days.

Vacuum Filtration for Slime.

The slime from the classifiers is thickened in Dorr continuous thickeners, but no attempt is made at counter-current washing. Air lifts are used for the transfer of pulp from one tank to the next, and gentle air agitation is used in the storage tanks to keep the slime pulp from settling. As stated above, the solution decanted from the thickeners is used as the first leaching solution on the sand tanks. The total time of slime treatment is about 36 hours, at the end of which time the pulp has a consistency of one part solids to one and a half parts solution, and is ready for treatment in the Moore vacuum filter.

At times difficulty is experienced in washing soluble gold from the slime cake, and this has led to further treatment of the slime prior to its final discharge. It has been observed that, although the gold was not readily washed from the cake, nevertheless it seemed to diffuse rapidly through the water in the unloading tank after the cake was blown off the leaves. The liquor in the unloading tank thus became too valuable to be wasted. Accordingly the usual practice has been modified, and before the discharged cake is finally run to waste it is thoroughly mixed with the solution in the unloading tank and pumped to circular, conical-bottom tanks, in which are suspended filter leaves made of wooden frames covered with poultry netting and canvas. No vacuum is applied, but a clear solution worth from thirty to fifty cents per ton is decanted, precipitated separately from the regular gold solution, and allowed to run to waste. The slime settles to the bottom of the tanks and is discharged periodically as long as it runs thick.

Precipitation is accomplished in the usual manner on zinc shavings. The gold solution has a value of $2 per ton, and the zinc-box tailing about two to four cents per ton. Sodium cyanide is used in the mill solution, which has a strength equivalent to 1.5 lb. KCN per ton. Consumption of chemicals per ton of ore treated during the last fiscal year was as follows: NaCN, 0.572 lb.; CaO, 8.956 lb.; Zn, 0.681 lb; and acid 0.248 lb.

Settlement with the different leasers is made on the basis of extraction results. A sample of one kilo is taken from each carload shipped during the month, and this sample is subjected to tub tests to determine percentage extraction

Oil-Fired Roaster at Astoria Mine.

Ore from the Astoria mine, owned by the Golden Reward company, is wholly unlike that of other mines in the district. It contains an average of 10 per cent sulphur, and in the raw state yields only 18 per cent or 20 per cent extraction by cyanidation. Preliminary roasting, however, greatly improves both the chemical and physical condition of the ore, and makes it amenable to cyanide treatment. With a view to treating this ore on a large scale, a roasting plant has been erected at the mine, consisting of a Wedge 20-ft., 7-hearth, mechanical roasting furnace, with the necessary complement of crushing, drying and cooling apparatus. The furnace is adapted to burn oil, having six burners disposed at equal intervals on the hearth next to the bottom.

Only one test run of about 75 tons has been made, using some dump ore which was not as favorable to good results either in roasting or cyaniding as freshly mined ore; nevertheless an extraction of 75 per cent was made when the ore was crushed to ¼-in. size, roasted and treated unclassified in leaching tanks. The furnace showed a capacity of 100 tons per 24 hours, reducing the sulphur content of the ore from 8 per cent to 0.2 per cent. Oil consumption was 9 gallons per ton of ore, at a cost of 3.9 cents per gallon. The temperature on the fuel hearth was 1300 deg. F., and about 300 deg. less on each succeeding upper hearth. The plant has demonstrated satisfactorily the feasibility of producing a roasted ore that will be readily amenable to cyanidation; and the continued success of this experiment on the part of the Golden Reward company is expected to be of material benefit to the industry in the Black Hills.

Further Roasting Experiments.

Since the completion of the large-scale test, further experiments have been continued on a laboratory scale to determine the effect of different temperatures and of a water wash prior to cyaniding. The tabulations below contain details of a typical roasting-cyaniding test, from which some general deductions can be drawn. It appears that a low-temperature roast, followed by a water wash, gives higher extraction of gold and lower consumption of lime than a high-temperature roast without preliminary washing. The consumption of cyanide does not vary greatly in either case. The extraction is good on both coarse and fine portions, though slightly better on the latter, and in less time. The ultimate selection of a method will depend on the cost of the various operations. The low temperature roast will require less fuel, and the time will be reduced, but the expense of washing will have to be taken into consideration.

Experiments also have been made by grinding the entire ore, coarse and fine, in a tube mill in cyanide solution. Portions were removed successively for treatment by air agitation and settling. The results of such a test are shown in the table below. A striking feature to be noted is that the percentage extraction is high on the 2-hour treatment, falling off sharply at 4 hours, and then gradually rising with increased time until at 51 hours it is the same as at 2, reaching its maximum at 74 hours. This feature was noted in several tests, and shows the ready solubility of the gold. The increased extraction at 74 hours would in no way compensate for the time and expense involved nor for the consumption of KCN and CaO, which also increased steadily with the time.

The consumption of KCN ranged from 0.3 lb. to 0.7 lb., and of CaO from 14 to 17 lb., increasing steadily with the time.

Still other experiments are now in progress on the hard blue ore, to determine the effect of roasting prior to cyaniding. This

PERCOLATION OF COARSE PORTION—SIZE, ¼-INCH

Temp. of Roast Deg. F.	Washed or Unwashed	Value		Consumption, Lb.		Extraction Per Cent	Time, Hrs.
		Head	Tail	KCN	CaO		
800	Washed	$15.10	$1.60	0.55	17.26	89.41	144
900	Washed	15.20	1.80	0.48	16.87	88.16	144
1000	Unwashed	15.20	2.40	0.38	23.52	84.21	144
1100	Unwashed	15.15	3.40	0.38	22.99	77.56	144
1200	Unwashed	15.20	3.60	0.56	19.08	76.32	144
AGITATION OF FINE PORTION—SIZE, 80 MESH							
800	Washed	15.10	2.00	0.30	9.12	86.76	16
800	Washed	15.10	1.50	0.50	8.66	90.06	48
800	Washed	15.10	1.50	....	.....	90.06	120
800	Washed	15.10	1.40	....	....	90.74	120
900	Washed	15.20	2.50	*2.20	21.21	83.55	24
1000	Unwashed	15.20	3.30	0.31	17.38	78.29	24
1100	Unwashed	15.15	3.60	0.20	17.82	76.24	24
1200	Unwashed	15.20	3.55	0.20	16.84	76.65	24

*An unusual result: 2.10 lb. KCN and 20.92 lb. CaO consumed in first hour of treatment.

TUBE MILL TEST ON WHOLE ROASTED ORE
Solution: 2 lb. KCN and 20 lb. CaO per ton

Successive Portions	A	B	C	D	E	F	G	H	I
Hrs. tube milling	1	1	1	1	1	1	1	1	1
Hrs. first standing	1	1	1	1	1	1	1	1	1
Hrs. first agitation	...	1	2	3	4	4	4	4	4
Hrs. second standing	...	1	1	1	16	16	16	16	16
Hrs. second agitation	...	...	...	...	...	4	4	4	4
Hrs. third standing	...	...	...	...	...	1	20	20	20
Hrs. third agitation	...	...	...	...	...	...	...	4	4
Hrs. fourth standing	...	...	...	...	...	...	...	1	24
Hrs. total time	2	4	5	6	22	27	46	51	74
Value of original ore	$11.70	thr	ough	out					
Value of tailings, $	1.90	2.30	2.20	2.20	2.20	2.10	2.05	1.90	1.80
Extraction, %	83.76	80.35	81.20	81.20	81.20	82.05	82.48	83.76	84.62

ore contains only 2 per cent to 3 per cent sulphur, but is nevertheless very refractory, and it is possible that the physical effect of the roast will prove beneficial in the cyanide treatment. The Golden Reward company is doing pioneer work in this branch of ore treatment in the Black Hills, and the ultimate outcome is of great moment and interest to the district. Present indications are that roasting may become recognized as a valuable adjunct to cyaniding, and that the practice may be extended as a result of the success attending the work of the Golden Reward company.

The Trojan Mill.

Ore treatment similar to that at the Golden Reward is practiced at the Trojan mill. The company operates a number of mines and treats a mixture of oxidized and unoxidized ores, including some blue ore. The average value of the mill feed is about $6 per ton, and the extraction averages 75 per cent.

Ore is gathered from the several mines by gasoline motor and delivered to three mill bins, which are so disposed and constructed as to discharge over grizzlies to the same gyratory crusher. The grizzly undersize and the crushed product are combined and elevated to the mill by a belt conveyor set at an angle of 16 deg. At the discharge end of the belt is a sampler similar to that used at the Golden Reward, consisting of buckets attached to a chain, designed to travel through the stream of ore falling from the belt. It will be apparent that the efficiency of this type of sampler depends on the delivery of a continuous, uniform stream of ore. Owing to the fact that in this case the feed to the belt is irregular, due to sticky ore and other causes, the sampler is unreliable, and its use has been discontinued. The mill results are checked by the value of bullion recovered and tailings discharged.

Roll Crushing in Solution.

The Trojan is one of two mills in the Black Hills where roll-crushing in cyanide solution is practiced, and its use here cannot be said to be satisfactory. The pulp passing the rolls flows to 7-ft. Chilean mills, where it is ground to pass ton-cap screen having slots 0.021 in. wide. These Chilean mills have extra wide rings and dies which are very satisfactory on soft ore, but not particularly desirable for hard ore. Solution is added at the rate of 4 to 6 tons per ton of ore, the quantity varying with the character of ore under treatment, more being used for soft ores that produce considerable slime, and less for hard ores. The mills have a capacity of about 225 tons each per 24 hours, or a total capacity of 450 tons for the whole plant.

The ground pulp is elevated to two Dorr classifiers working in series, the sand discharged from the first being reclassified in the second. The separation yields about 55 per cent sand and 45 per cent slime. The sand is delivered to leaching tanks through the ordinary distributers, where it is leached with strong solution for three days and barren solution for two days, after which it is given a water wash and discharged in the usual manner. Extraction in the sand tanks is not as good as in the slime department. The sand tailings have an average value of $1.60 per ton.

Slime Treatment.

The slime is thickened in a Dorr continuous thickener to a consistency of 1½ parts solution and 1 part solids, and then subjected to air agitation in two tanks in series. Several forms of central tube have been tried in the agitators, including long and short tubes, and a form comprised of a series of hollow truncated cones connected together. The object of the latter type of central tube was to permit discharge of the pulp at successively higher points as the tank filled with slime. Apparently, however, equally good results have been obtained without central tubes, and they have been removed entirely from some of the tanks.

Vacuum filtration is practised with a form of stationary leaf filter. A smooth, uniform cake of 1 in. thickness is readily formed in about 50 minutes. This is washed with barren solution for 70 minutes, and with water for 12 minutes before discharging. As at the Golden Reward, some difficulty is experienced in washing soluble gold from the cake in reasonable time. The washed slime tailing assays about $1.10 per ton, which is 50 cents lower than the sand tailing.

Recovery of Water from Tailings.

All tailings are impounded by a sand dam, below which is a bed-rock concrete dam. In this connection it is interesting to note that the water, or weak solution, recovered at the dam contains gold to the value of 15 to 20 cents per ton, representing probably soluble gold remaining in the slime and sand tailings. It is planned to recover this solution after it has percolated through the sand dam, and precipitate it on charcoal, after which it will be pumped back to the mill for further use.

The gold solution from sand tanks and slime filter is precipitated on zinc shavings in the usual manner. The head solution assays $2 per ton, and the tailing from 1 to 1½ cents. The strength of the mill solution is 1.5 lb. KCN, with protective alkalinity of 1.5 lb. CaO. Chemical consumption is 0.6 lb. KCN, and 8 lb. CaO per ton of ore. The gold precipitate is cleaned up, acid-treated and melted in the usual manner.

Proposed Changes and Improvements.

The Trojan mill will be remodeled this summer, and its capacity increased to 500 tons per day. Wet crushing at the rolls will be discontinued, and solution will be added to the roll discharge to convey it to the Chilean mills. The latter will be fitted with screen of larger openings, ton-cap 0.040-in. slot, and a short tube mill will be installed to regrind about 40 per cent of coarse sand. Experiments on Trojan ores indicate that grinding to minus 60-mesh gives the most efficient results. Finer grinding is without material benefit. Hence, it is planned to classify the Chilean discharge to give a sand containing practically all the plus 60-mesh particles for tube milling. The tube will work in closed circuit with the classifier, the overflow from which will then be elevated to a series of Dorr classifiers for separation into fine and coarse products. The present practice of adding dry lime at the crushers will be dis-

continued, and slaked lime will be added at the Chilean mills.

Steel-housed belt and bucket elevators are being built to raise the ground pulp to the Dorr classifiers. The steel housing is sectional, and the plates are riveted except at those points where the occasional removal of a plate is desired. In the latter case the plates are bolted in place, the bolt heads being inside and held in place by a strip of grooved metal. By this construction a plate can be readily removed without inconvenience arising from the displacement of the bolts. The boot pulley has outside bearings, and the shaft runs through split hardwood bearings placed in the housing to prevent leakage at that point. The boot is provided with an overflow just below the shaft, and with a discharge pipe at the bottom through which accumulations can be sluiced to a sump.

Counter-Current Washing to Be Adopted.

Additional thickeners and agitators will be installed, and a system of continuous counter-current washing and agitation will be employed. The details of this treatment have not been worked out, but it is planned to decant solution continuously for precipitation, and to agitate in fresh solution after thickening. By this scheme it is hoped to wash the valuable solution from the pulp before it reaches the filter, and to use the latter primarily as a dewatering machine.

A Lea V-notch recorder, the first to be used in the Black Hills, will be used as a gold solution meter. The Trojan management is of the opinion that this form of solution meter will prove superior to any form of enclosed meter or oscillating boxes, owing to the difficulty of preventing the deposition of scale in the former, and of properly standardizing the latter.

Dry Crushing at Wasp No. 2.

As an example of low cost treatment of a very low grade ore, Wasp No. 2 mill holds an unique position in the Black Hills, and indeed is almost without parallel in the realm of cyanidation[4]. The ore is coarse grained quartzite, sufficiently porous to permit ready solution of the gold and a high rate of leaching when crushed to pass a ¼-in. mesh screen. According to the annual report for 1912, the ore averaged $2.34 in gold and 0.64 oz. silver per ton. The extraction was 75.9 per cent of the gold and 30.3 per cent of the silver. Under these conditions, and with a daily capacity of 500 tons, the profit on the operation is about $5,000 per month. Tonnage is a factor in the profitable operation of this mill; at a capacity of 300 tons per day, a net loss would be sustained, while at 400 tons per day expenses would be paid, but no net profit would result. The total mining and milling cost for 1912 was $1.2337 per ton, of which 62.65 cents was for milling. The distribution of this item was as follows:

	Cents per ton
Labor	20.57
Supplies	6.00
Repairs	9.10
Coal	0.85
Cyanide	8.06
General	0.62
Stable	0.90
Assay Office	0.69
Superintendence	1.58
Lime	1.36
Clean-up	0.87
Zinc	3.64
Power	8.28
Tools	0.13
Total	62.65

Crushed Ore Not Classified.

The system of treatment is extremely simple as compared with that necessarily adopted where sand and slime must be separated. The ore is crushed in gyratories, first to 2½ in. and then to 1½ in. The latter product is further reduced in roughing rolls, and passed over stationary, inclined rek-tang screen of ¼-in. mesh, the oversize from which is returned to finishing rolls. Dry lime is added at the second gyratory, which is a practice that may be criticized on account of the coarse condition of the ore and the improbability of thorough mixing in the storage bins.

The crushing faces of the gyratories and rolls are made of manganese steel. Gyratory mantles last about one year, and concaves six months. Roll shells wear for 50 days. These figures relate to the treatment of 500 tons per day.

No screen analysis is available on the product passing the screen; but although it contains more or less fine sand, percolation and extraction proceed at a rapid rate. The ore is loaded into 32-ft. by 12-ft. leaching tanks of slightly over 400 tons capacity, by means of a system of conveyor belts. As the ore drops into the tank it is distributed by shoveling to avoid classification of coarse and fine particles. Twelve tank charges are treated every ten days.

The cycle of sand treatment is as follows: A 5-lb. cyanide solution is run onto the charge until the ore is covered. Contact is continued for 12 hours, after which the tank is drained and the contents treated with successive applications of a 2½-lb. solution for from 48 to 56 hours. A water wash completes the treatment. Strong and weak solutions are precipitated separately on zinc shavings, the head and tailing values of the strong solution being $3 and $0.08, and of the weak solution $1 and $0.02, respectively. One ton of solution is precipitated per ton of ore treated. Chemical consumption per ton of ore is: KCN, 0.35 lb.; CaO, 5 lb.; Zn, 0.3 lb.

Cost of Unloading Tanks.

Owing to the situation with relation to the railroad, it is impossible to discharge the tanks by sluicing, and the tailings have to be stacked dry. Shovelers unload the sand through gates in the bottom of the tanks, filling cars which are then trammed to the dump. A system of belt conveyors might be used for this purpose were it not for the difficulty of operating in winter. The cost of unloading is about five cents per ton, a tank being discharged in seven hours by seven men, four of whom are engaged in shoveling and three in tramming.

The gold precipitate from the zinc boxes is recovered and treated in the usual manner, producing bullion of a total fineness of 900 to 930.

Owing to the uniform character of the ore and the simple system of treatment, it is considered unnecessary to use automatic sampling devices for arriving at the tonnage and value of ore treated. Each tank charge is pipe-sampled at various points, and the estimate of tonnage and value made in this way.

For courtesies extended and information furnished for these notes, acknowledgment is due to Messrs. Henry Schnitzel, F. R. Baldwin and M. E. Hiltner, of the Golden Reward company; H. S. Vincent and Charles Ellis, of the Trojan; and John Gray and C. E. Brenner, of the Wasp No. 2.

[4]An example of coarse-crushing and leaching in Mexico was described in this journal, May, 1913, p. 291.

Absorption and Reaction Towers for Chemical Factories.—By a mistake the name of the author, Mr. **Rudolph Heinz**, of Germany, was omitted in the publication of the article on page 359 of our June issue.

The Aluminium Industry Co., of Neuhausen, Switzerland, reports a net profit of $717,000 for 1912, as against $448,000 for 1911. It is stated that the low prices which prevailed for some time influenced the development of the industry by creating new uses and demands for the metal. A new agreement has since been entered into by the various aluminium makers.

Thermit marine repairs is the title of a very interesting and well-illustrated pamphlet of 30 pages of the Goldschmidt-Thermit Company, 90 West Street, New York, giving an account of recent progress made in marine repair work with the use of the thermit process with various examples from American practice.

The Purification of Blast Furnace Gases.

A Report of a Paper by C. Herwegh, Giving Data on the Use of the Feld Washer in French Blast Furnace Practice—Details of a Proposed American Installation

In our issue of July, 1912, we published an article by Mr. Walter Feld on his centrifugal gas washer and its application for the purification and cooling of blast furnace gas and producer gas and other purposes.

In this connection a paper by Mr. **Camille Herwegh,** read before the Mining and Metallurgical Section of the (French) Société Industrielle de l' Est is of considerable interest as it gives data from actual practice at the *Pompey blast furnace plant* which has been the first iron blast furnace plant to adopt the Feld washer on a large scale.

The author first gives data on the quantities of blast furnace gas available and its usual calorific value and the troubles due to the content of dust in the same. In their furnaces "de l' Est" the quantity varies from 8 to 18 grams per cubic meter of gas (3.52 to 7.92 grains per cubic foot). The obnoxious influence exercised by this dust is readily understood; it adheres to the walls of the recuperators and to the flues of the boilers, thus forming a coating which very greatly diminishes the efficiency of the apparatus. This demands frequent cleaning of the gas passages and of the various apparatus; the dust in the gas therefore produces a perceptible loss in fuel and a higher maintenance cost. If used in the engines, the dust packs in the valves, thus in a short time preventing their proper operation.

Methods of Purifying Gas

Mr. Herwegh then gives the following outline of different methods of purifying the gas.

"A great many systems were both proposed and tested for this purpose; the first ones were constructed of channels having many changes in direction and area, or of compartments with partition walls located in different directions. This was the 'dry' method of purification, and it is still in use for precipitating the heavy dust, which falls down easily, but it is not sufficient for all purposes. To obtain a more complete cleansing, columns of coke or gravel with a continuous flow of water were used, this being the beginning of 'wet' purification, but the results left much to be desired.

"The gases passed through this apparatus without proper contact with the water, and were purified until they contained about 3 to 5 grams of dust per cubic meter (1.32 to 2.2 grains per cubic foot) at the outlet. Afterwards coolers made their appearance; they at times had enormous dimensions, and used large quantities of water with a consequent high amount of motor power.

"Before we erected our purifying plant at Pompey we tried primary purification without the use of water or power. This apparatus, designed by M. Pérard, mining engineer, was so constructed that the gas passed through pipes around which air was circulated—the air being continuously renewed by natural draft. It was provided with a dump for dust and for condensed water. This apparatus, which is still being used for cleansing the gas used under the boilers, necessitates large dimensions to enable it to purify 3,000,000 cubic meters (105,900,000 cu. ft.) per twenty-four hours.

"In the coolers using water circulation, the gases having a temperature of 80° C. (176° F.) at the inlet, are cooled down to 30° C. to 25° C. (86° to 77° F.), a certain quantity of dust is precipitated and a large portion of the water is condensed; the efficiency, however, is very low compared with the amount of water used—from 3 to 5 liters per cubic meter (0.09 to 0.15 quarts per cubic foot)—and the gas still contains from 2 to 3 grams of dust (0.88 to 1.32 grains per cubic foot) at the outlet.

"Ventilators were placed behind this apparatus, and here from 1 to 2 liters of water per cubic meter (0.03 to 0.06 quarts per cubic foot) were injected, and we were thus able to effect the desired degree of primary purification; that is down to 0.05 to 1 gram (0.22 to 0.44 grains per cubic foot) for the gas used in the recuperators and boilers. But these ventilators absorb a great deal of power. For instance, a ventilator passing 50,000 cubic meters (1,765,000 cu. ft.) of gas per hour without water injection absorbs 50 hp, while the same ventilator absorbs 180 hp when 2 liters (0.06 quarts per cubic foot) are injected.

"There are a great number of patented washers designed to replace coolers and ventilators with water injection, but none of these have heretofore operated with much success.

Feld System

"This system effects the cleansing and cooling consecutively in such a manner that the primary cleansing is completed before the gases are completely cooled down.

"The apparatus consists of a washer (Fig. 1) built up of sections one above the other. The gas inlet and the outlet are indicated in the diagram; the gas passes upwards from one washing chamber to the other through gas ports.

"Each section is provided with a water-spraying device at-

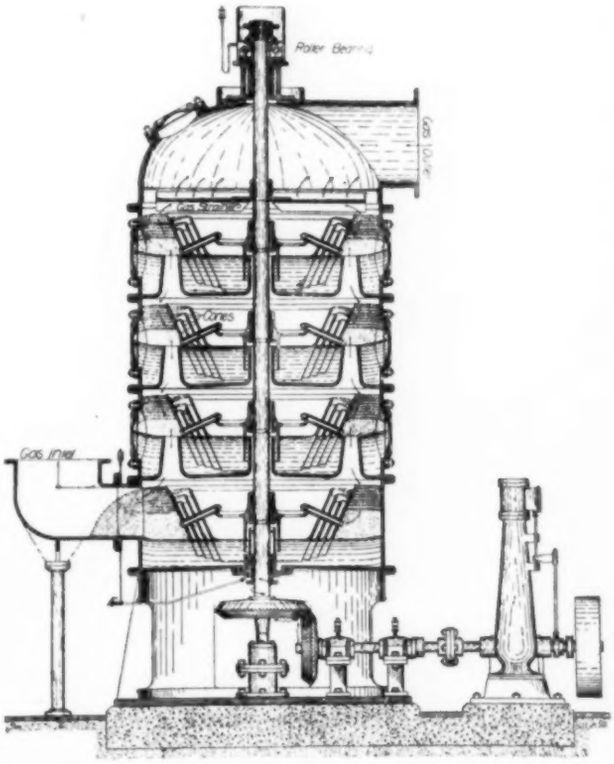


FIG. 1—FELD GAS WASHER

tached to the vertical shaft. This spraying device is composed of a series of concentric cones attached to a hub keyed to the shaft, the lower ends of the cones being submerged in water. By revolving the shaft, and hence the cones, the water is raised by centrifugal power along the inner sides of the cones and atomized at the upper edge.

"The upper edge of each cone is a little higher than the next outer one, thereby forming, according to the number of cones, a certain number of horizontal sprays of water. The upper portion of the outer cone, which is somewhat higher than the inner one, is perforated, the inner cones supplying water to the perforated surface of the outer one. This forms a series of cascades composed of excessively small drops of water through which the gas to be cleansed must pass. The water is violently thrown against the walls of the washer, and then returns under the revolving cones; this permits of using the same body of water as long as it may be desired. Theoretically, only such amount of water flows out of each section as may rise above the normal level. The washing is done in the lower sections while the cooling takes place in the upper ones.

"In order to decant only a small portion of the dust-bearing water, about three-quarters of the water is taken from the

third section from the top, without thus decreasing the cleansing, or the efficiency of the Feld washer is such that the wash water may be concentrated without detrimentally influencing the purification. It is also noted that hot water washes the gas much better than if cold water were used, and all sediment is thereby avoided.

"The upper part of the washer is provided with perforated plates, which, through their action, cool the purified gases above the cones. This cooling plays an important role in the condensation of the water vapor. In the later washers these perforated plates are replaced with advantage by another series of cones. The washer has a diameter of 2.9 meters (9 ft. 6 in.) and a height of 5.8 meters (19 ft.).

"You can readily see that this washer acts both as a purifier and as a cooler; the ventilators, which they have replaced as purifiers, now only serve to draw the gas through the apparatus and thus to overcome the resistance offered in the pipe connections and in the washer itself. I must state, however, that a small additional quantity of water must be injected into the ventilators to prevent the slow precipitation of dust which still remains in the gas at the outlet of the Feld washer and which dust would adhere to the inside of the ventilator.

"Washers of the same construction are used for the recovery of tar, tar oils, benzole, cyanogen, ammonia and hydrogen sulphide from gas.

European Installations

"The plant at Engers, Germany, tried the Feld washer in a small way in connection with a blast furnace; there was only one washer, but it was not connected with a ventilator, and only such amount of gas as happened to pass through the washer was purified.

"The lead mines at Mechernich-on-the-Rhine, where they work up a 4 per cent lead ore, also erected two washers to recover the lead dust which formerly accumulated in five immense galleries placed underground and leading to the stack. These galleries were built in sections of 400 meters (1316 ft.) each, or 2 kilometers (1.25 miles) in all, the chimney having a height of 130 meters (427.7 ft.). In spite of these galleries, the gases leaving the chimney inundated the surrounding country with their unhealthy dust contents, which were excessively rich—containing 80 per cent to 100 per cent of lead. Twice a year, consuming a period of three weeks each, the entire plant was shut down to permit of cleaning out these galleries by means of wheelbarrows and carts. You can readily understand how unhealthy this work was to the laborer, and the loss sustained by the company during these cleaning periods, as well as by the damage done by the gases escaping from the chimney.

"In order to prevent this loss, two Feld washers were erected, each with but two sections, working in series, a ventilator being attached to the second washer.

"During twenty-four hours this apparatus recovered six tons of dust, containing 80 per cent to 100 per cent of lead. From these tests in the lead mines, where the dust was entirely recovered, we were positive that this apparatus would work satisfactorily if applied in a large installation for the cleansing of blast furnace gases and for condensing the water vapor carried in the gas.

Power Consumed by the "Feld" Washer and Other Details

"An apparatus such as I am about to describe to you, having a capacity of 1,000,000 cubic meters (35,300,000 cu. ft.) of gas in twenty-four hours, consumes only from 15 hp to 20 hp. The shaft is suspended in a ball bearing. When the motor is stopped the shaft continues to revolve for about twenty minutes, the water continuing its circulation. The washer consumes about 3 liters of water per cubic meter of gas (0.09 quarts per cubic foot).

"After several months of operation the apparatus was stopped to be inspected and cleaned, if necessary, but no sediment was found and the washers were closed again. During a period of eighteen months the washers at Pompey were run continuously—day and night—and were never closed down for repairs.

"The degree of purification is from 0.4 to 1 gram of dust per cubic meter (0.17 to 0.44 grains per cubic foot). The gases leave the blast furnace carrying 120 grams of water vapor per cubic meter (52.8 grains per cubic foot) and leave the Feld washer with 2.8 grams (1.23 grains per cubic foot) above the theoretical point of saturation corresponding to the temperature.

Primary Purification Plant at Pompey

"When the gases leave the furnaces they first pass through the 'dry' purifiers; at the outlet the gases have a temperature of 150° C. (302° F.) and contain from 12 to 15 grams of dust (5.28 to 6.6 grains per cubic foot) and 120 grams of water vapor (52.8 grains per cubic foot) per cubic meter. They enter at the bottom of the first 'dry' dust catcher, this having a diameter of 3 meters (9.87 ft.) and a height of 9.5 meters (31.26 ft.) over the straight shell; they leave at the top through a pipe having a diameter of 2 meters (6.58 ft.) and enter the bottom of the second 'dry' dust catcher having a diameter of 4 meters (13.16 ft.), leaving at the top and entering a general collector 2.8 meters (9.21 ft.) in diameter. You can see from these dimensions that in the 'dry' dust catchers we have combined both a change in direction and a change in cross-section of conduits, so as to vary the velocity of the gases and to facilitate the precipitation of the dust. The gases do not contain more than from 3 to 5 grams of dust per cubic meter at the outlet (1.32 to 2.2 grains per cubic foot).

Wet Purification

"Contrary to the usual practice at several large plants, where only the gas required for gas engines is purified, the entire gas coming from the furnaces at Pompey receives primary purification, and the benefit thus realized in the increased efficiency of the boilers and cowpers, by reducing the cost of cleaning and maintenance, is very important.

"The entire gas from the furnaces, which is collected in the general collector, is passed through the Feld purification plant. This plant consists of three washers, each washer being connected to an independent ventilator; the plant is erected within a building 12 by 16 meters by 15 meters high (39.5 x 52.64 x 49.4 ft.). The apparatus was supplied and erected by the well-known firm of "Fives Lille," which has acquired the French patents. The motors and shafting operating the washers, as well as the pumps, are located on the first or basement floor, the washers and ventilators being placed on the second floor. A traveling crane above the washers permits of quickly dismantling and removing any part of the apparatus. The ventilators force the gas either to the cowpers and boilers, or to the final cleansing plant for the gas engines.

"A special main was provided to carry the raw gases direct from the collector to the ventilators, without passing through the washers. If this is used, the ventilators must be supplied with an additional amount of water for cleansing the gas if the washers are by-passed for repairs or any other purpose. This was a very important precaution in trying out a new system, but it has never been used.

"Below are given the average results taken from numerous tests made during a period of three months, or from December, 1910, to February, 1911:

"Quantity of dust per cubic meter of washed gas at 0° C. and 760 mm. pressure (32° F. and 30 in. pressure)—0.483 grams (0.21 grains per cubic foot).

Temperature of the washed gas above the water in the washer cooler—6.3° C. (43.34° F.).

"Quantity of water in the washed gas above the theoretical point of saturation, corresponding to the temperature—2.83 grams per cubic meter (1.245 grains per cu. ft.).

"Water consumed per cubic meter of gas at 0° C. and 760 mm. pressure—3.62 liters (27.05 gallons per 1000 cu. ft. of gas.)

"The impure gas contained 121.6 grams per cubic meter (53.5 grains per cubic foot) of water vapor at the inlet; if it had been 80 grams (35.2 grains per cubic foot), on which the guarantee was based, the amount of water would have decreased perceptibly.

"The three washers consumed 15 hp each. The three ventilators, which, in order to maintain a difference in pressure, consumed 35 hp each, would have consumed from 60 hp to 70 hp each if operated with a difference in pressure of 160 mm. (6.3 in.) between inlet and outlet. (The power consumed by shafts and belting has not been included in the above.)

Decantation

"The decantation process is also well worthy of description.

"The dust-laden water is carried by an underground canal to a concrete reservoir, from where an electric pump forces it up to a pointed wooden vat known as "spitzkasten" and used in coal washeries. The water enters at one end of this vat, and the openings for distributing the decanted water are at the other end. Broom straw, placed on supports at the water level, hold back the scum and facilitate decantation. The heavy-laden water accumulates at the bottom and is drawn off by means of goose-neck-shaped pipes—the pipes being provided with cocks at their lower end—into a concrete basin; the sediment being drawn from the bottom of this basin into cars and removed to a dump."

Proposed American Gas Washer Plant for Two Blast Furnaces Producing 5,400,000 Cu. Ft. of Gas per Hour

In connection with this paper the following design of a Feld gas washer plant of a capacity sufficient for two blast furnaces producing 5,400,000 cu. ft. of gas per hour is of special interest. For this information we are obliged to Captain **Fred H. Wagner,** chief engineer of the Bartlett Hayward Company, Baltimore, Md. This company owns the patents for the Feld washer in this country.

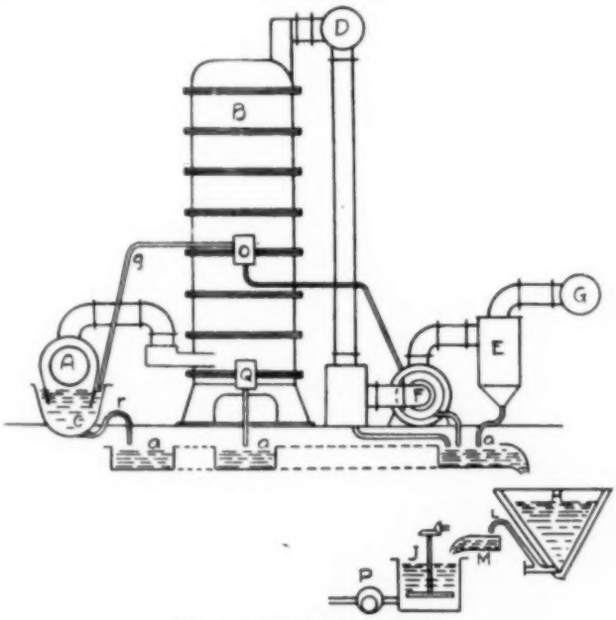


FIG. 2—GAS-WASHING PLANT

In the plant, as further described below, the necessity of arranging the apparatus in such manner that it occupies as little space as possible is of prime importance, at the same time bearing in mind the necessity of so arranging the apparatus that a general supervision and easy access to each individual piece of machinery is possible.

Design for Continuous Operation

Furthermore, in contradistinction to all of the usual plants seen in America, this one is arranged to avoid all periodic interferences with continuous operation.

As usually constructed at present, we find the plants subject to numerous cleaning periods for the gas mains as well as for the washing apparatus, that is, the removal of accumulated dust, which periods entail serious interruption to continuous operation. This condition has been completely overcome by the arrangement adopted for both the washers and the pipe connections; also the periodical cleaning of the settling basins, entailing a considerable expenditure of labor and money, is completely obviated, due to the method adopted to automatically concentrate the mud in the clearing basins, and the continuous disposal of the concentrated mud by mechanical means, thus reducing the labor to a minimum.

The entire plant is placed at a sufficient elevation above the ground line to permit of the overflows from the various machines to empty into the clearing basins, the cleared water to run into a sewer and the overflows from the clearing basins to empty into the mud concentrators and from there to the mud pumps by gravity, all pumping inside the plant being thus avoided.

The apparatus is designed to primarily clean the gas from two furnaces, the quantity of gas from each furnace amounting to 2,700,000 cu. ft. per hour (76,487 cbm.), or 45,000 cu. ft. per minute (1275 cbm.) at 250° F. (120° C.), and, as shown in Fig. 2, it consists of:

A = Inlet main from dry dust catcher.
B = Vertical centrifugal washer.
C = Hydraulic main.
D = Collecting main.
E = Water separator.
F = Exhauster.
G = Primary gas main.
H = Cone basin, or clearing basin.
J = Mud concentrating tank.
P = Centrifugal mud pump.
r = Mud overflow.
a = Mud trough.
O = Proportional overflow.
Q = Washer overflow.
L = Mud overflow from settling tank.
M = Mud trough.
g = Water to main.

The unpurified gas passes from the furnace through the down comer into the dry dust catcher (not shown) and from thence by means of the hydraulic main A into the washing plant. In order to avoid periodical cleaning of the gas main from the dust catchers to the washers B, the horizontal stretches of the main A are formed as a dip, the lower open edges of the main dipping into the water seal C. The dust, which may be separated from the gas by friction during passage through this main, is deposited in the trough C, which is always filled with warm running water, and the resultant mud in a concentrated form is automatically carried off through the syphon overflows *r* to the mud trough *a*, and from thence to the cone basin H, where the mud receives a further degree of concentration.

A reduction in the cross-section of the inlet pipe by the deposit of dust or mud on the inner surface and a consequent loss in pressure is thus avoided; the periodic interruption caused by necessary cleaning, as well as the usual disagreeable condition produced by the dust during the period when the usual type of inlet main is being cleaned is also avoided. Besides this, the hydraulic main provides a sufficient safety valve during a possible explosion in the furnace.

The gas is now admitted into washer B, where it is cooled and cleaned, the dust content being reduced to 0.13 or 0.20 grain per cubic foot, dependent upon conditions, and the gas being cooled to about 77° F., the gases entering the washer with a temperature of 250° F. and the water vapor content of 53 grains per cubic foot; it is possible, however, that the gases may enter the washer at a higher temperature, and the higher the entrance temperature the better will be the effect of dust washing.

At the inlet nozzle of the washer the hot, saturated gases come into intimate contact with the hot water in the lower washing chamber, whereby the temperature of the gas is brought to the equalizing temperature in this case to about 57° C. = 135° F., and the unit volume is thereby reduced. The washers are provided with three lower washing chambers, one intermediate or separating chamber and three upper or cooling chambers.

The three lower chambers are supplied with hot water, this water having been heated in the upper cooling chambers and the temperature of which changes with the total heat value of the gas, depending upon the water contents and the entrance temperature of the gas, this hot water being the active dust extracting medium.

In the upper or cooling chambers, the gas coming from the washing chambers with a temperature of about 57° C = 135° F. is cooled down to 25° C. = 77° F. The final temperature is, of course, dependent upon the total heat value of the gas and upon the temperature and amount of cooling water used. The quantity of water required for cooling the gas is admitted into the top cooling chamber, the water extracting heat from the gas during its passage through the chambers; about a fifth to a sixth of this hot water is admitted into the lower or washing chambers, where its temperature is increased from about 50° C. = 122° F. to 60° C. = 140° F.

The hot water coming from the overflow O under the cooling chambers, and which water contains but little dust, passes into trough C under the inlet main A and serves to produce a flow in the trough. This water finally passes into the cone basin H by means of the syphon *r* and mud trough *a*. The hot water which leaves the washer at Q contains the major portion of the dust washed out of the gas, or about 90 per cent, and it flows directly to the cone basin H.

The cooled and primarily cleaned gases now pass through the collecting main D and from thence down to the exhauster F and water separator E, which in turn delivers it to the primary gas main G, and the gas is now ready for use as fuel in the stoves, furnaces or boilers.

The exhausters are so constructed that it is possible to operate them with water injection, but normally only about 7.5 gallons of water coming from the overflow O would be run into the exhauster per hour in order to clean the casing and revolving vanes; but the water connections to the exhauster are of such diameter that the exhauster can be supplied with a greater amount of water from time to time if such should be required, so that the machine can be thoroughly cleaned without shutting it down.

If it should be desired to clean the gas for gas engine fuel, it would have to be subjected to final or minute cleaning. To do this it would be necessary to place another washer in the system, but this washer would not be provided with cooling chambers, and the connections should be so arranged that the necessary amount of engine gas could be diverted from main G into the final washer.

As stated above, the entire mud-bearing water is conducted into the cone basin H for concentration. Experience has shown that these cone basins are sufficient for the purpose, but if further concentration should be required this can be added

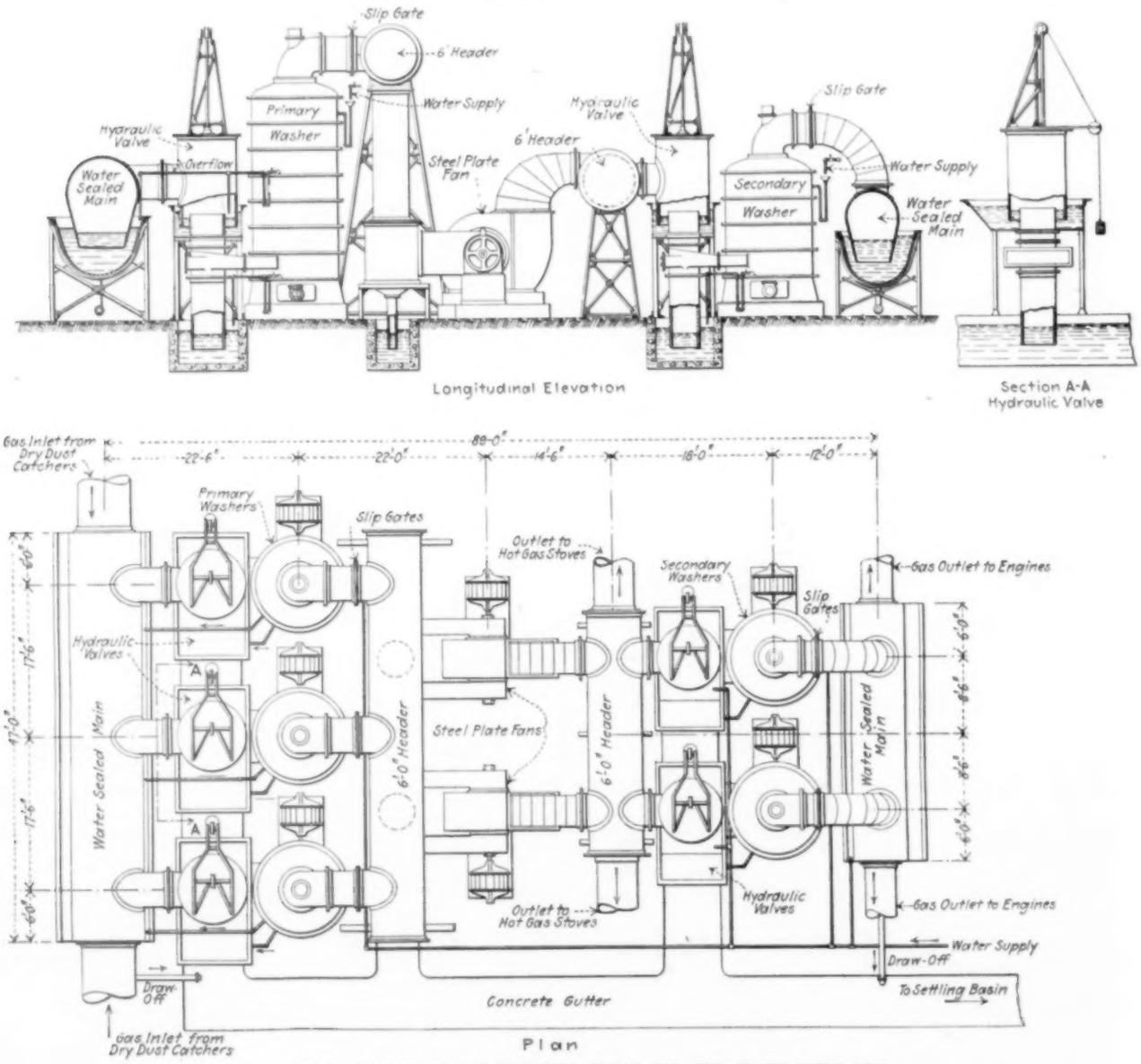


FIG. 3.—LAYOUT OF GAS-CLEANING PLANT FOR TWO BLAST FURNACES

to. The time required for clearing the water is dependent upon the nature of the dust and upon the temperature of the washing water, and therefore upon the temperature of the gases.

The mud-bearing water is distributed over the area of the basin by means of a distributing trough placed at one end; the mud, owing to the slow flow of the water through the basin, is deposited on the bottom, and the cleared water passes out on the other end and is conducted to the sewer.

The deposited and concentrated mud is automatically carried off by the overflows L into the collecting trough M, from whence it flows by gravity into the agitators and collecting tanks J. These agitators are supplied with centrifugal mud pumps P, and the outflow is so arranged to the pumps by gravity that they have no suction, their only requirement being to force the mud to the final depository. The collecting tanks J are connected with each other by means of overflows, the agitators being intended to prevent a settling of the mud during pumping periods.

Fig. 3 is a detailed drawing of the layout of a plant for two blast furnaces to be constructed by the Bartlett Hayward Company of Baltimore for an American blast-furnace plant.

Metering and Recording the Flow of Fluids*

NEW METERS AND RECORDERS OF THE FLOW OF LIQUIDS THROUGH VENTURI TUBES, ORIFICES, OR CONDUITS, BASED ON THE INTEGRATION OF THE VELOCITY HEAD

By J. W. Ledoux

The flow of water and other liquids as well as gases follows the law of gravity. For steady flow this may be represented by the expression: "Velocity is proportional to the square root of the head-producing velocity," or, algebraically,

$$V = C\sqrt{H}$$

in which

V = the velocity, in any convenient unit,

H = the head producing such velocity,

and

C = a constant depending on the units of measurement and the character of space through which the fluid flows.

V may be expressed in feet per second, miles per hour, etc., and H, the effective head, may be expressed in feet or in pounds per square inch, etc.

The space may be a pipe of any length, a submerged orifice, an open inclined channel, a Venturi tube, or a nozzle. It is seen that this is the form of the simplest equation of a parabola $y^2 = cx$.

For practical purposes, the ideal form of meter would be a device, connected by small tubes with adjacent parts of a pipe through which fluid is flowing, which would measure automatically the square root of the head or pressure producing the velocity. Such a device would offer no resistance or obstruction to the flow through the pipe. It has been impossible, heretofore, to achieve this ideal, because the forces due to velocity head are so minute as to be difficult, if not impossible, to measure or integrate when the velocity is low.

Theoretically, the Venturi tube is a contraction in the pipe, formed so as to cause small frictional resistance to flow; and it is remarkable how slight this contraction may be in order to produce the same difference in velocity head as that caused by Pitot tubes without contraction in the pipe. In fact, the diameter of the contracted portion need not differ from that of the full section by more than 17.7 per cent in order to produce the full effect of two Pitot tubes, one pointing up and one pointing down stream, with their openings in the center of the pipe, or at the points of maximum velocity.

In practice, however, the Venturi tube is composed of a short pipe continuously connected with the main pipe by a conical section pointing down stream, the convergence of these two sections being dependent on experimental determinations, the object being to obtain the highest flow with the least frictional or unrecoverable loss of head through the entire Venturi tube. These angles of convergence and divergence have been determined by Venturi and, in more modern times, by Weisbach, Eytelwein, Francis, Smith, Herschel, and possibly others, but at any rate it is pretty well settled that they should be somewhere between fairly wide limits, beyond which the results are certainly not as good.

However, they may be relatively the same for all Venturi tubes, whether the contraction is nearly as large as the pipe or very much smaller, the difference being that in the former case the entire Venturi tube would be short and in the latter long.

Of course, the purpose of a Venturi tube, the contracted section of which is small as compared to the full bore of the pipe, is to exaggerate the velocity head indications so that they can be observed and measured, especially for low flow.

To illustrate, consider the simple formula of a standard Venturi tube, having a coefficient of velocity = 0.977:

$$V = 7.84\left(\frac{H}{(D/d)^4 - 1}\right)^{1/2}$$

in which the only two variables are V and H.

V = the velocity through the full section, in feet per second;

H = the difference in piezometric heads between the up-stream section and the contracted section, in feet;

D = the diameter (or radius) of the full section;

d = the diameter (or radius) of the contracted section.

This formula may be written:

$$H = 0.0163\,[(D/d)^4 - 1]$$

Now, assume a velocity in the main as low as 0.25 ft. per second. Then the formula may be written

$$(D/d)^4 = 1 + 982\,H$$

and assume 0.01 ft. to be as low as it is possible to indicate H; then

$$D/d = \sqrt[4]{10.82} = 1.86,$$

which means that when the velocity through the main is as low as 0.25 ft. per second, if it is desired to get an indication of the velocity head as great as 0.01 ft., the contracted section of the Venturi tube must be less than 54 per cent as large in diameter as the main.

However, with a tube having ratios of diameters equal to 3 to 1, $H = 0.0815$ ft. for $V = 0.25$ ft. per second. Even this result cannot be obtained with the best apparatus formerly in use, unless one of the essential features of the Venturi meter is seriously sacrificed, namely, the measurement of high velocities.

The best types of Venturi meter registers have formerly depended for their registration on some form of cam having a parabolic curve, fashioned around a drum or revolving about a center. Near zero, or the origin, the curve changes in direction rapidly, and at zero crosses one of the axes of reference at right angles and becomes parallel to the other.

It is probably this feature, more than any other, which has limited the minimum velocity measurable by a Venturi meter; and though the measurement of low velocities is attainable by a high ratio of tube diameters, there is thus encountered the other difficulty of excessive frictional loss through the Venturi tube for comparatively high velocities.

If it is even intended to measure velocities as high as 10 ft. per second, which is occasionally desirable, the frictional head through a Venturi tube having a ratio of 3 to 1 in diameters, or 9 to 1 in areas, would be 17 ft., obtainable from the formula $H_f = 0.0021\,V_2^2$, in which H_f = the frictional head in feet and V_2 = the velocity through throat, in feet per second.

Besides that, in any form of register depending on mercury as the medium of indicating the velocity head, in order to cover ranges of velocity of from 10 ft. per second down to zero, the apparatus would have to be more than 11 ft. high. In a tube having a ratio of 2 to 1 in diameters, or 4 to 1 in areas, with a velocity of 10 ft. per second through the main, the frictional head would be only 3.36 ft. This is still a material loss, but a velocity of 10 ft. per second is unusually high.

*Extracts from a paper published in the May, 1913, issue of the *Proceedings of the American Society of Civil Engineers*, page 1065.

It must be remembered that, as far as metering a fluid is concerned, any form of orifice or contraction in a pipe would be just as good as a Venturi tube. It is also highly probable that the coefficient of flow would remain as nearly constant through just as wide a range of velocities; but the great advantage of a Venturi tube is in its small frictional resistance to flowing fluid, and in this respect it is the best shape yet devised.

The writer will now describe a form of meter register particularly adaptable to the registration of flows through any form of orifice, such as by the use of Pitot tubes and Venturi tubes of small ratios and consequently minimum frictional heads. The fundamental principles were discovered by the writer in 1904 and were finally developed into the best practical working form in 1909. The basic idea is a float, shaped so that, when acted on by the head due to the velocity of flowing liquid, its movement will be proportional to that velocity.

The device is novel in form, and covers a principle in hydraulics, which, although rational and perfectly obvious on analysis, the writer believes has not heretofore been recognized. This may be stated as follows:

If a hollow body of any shape, with open bottom (Fig. 1), floats freely in a liquid having the same specific gravity and touches another liquid of greater specific gravity, and if the relative pressures wthout and within this hollow body are changed so as to make the internal pressure less than the external pressure, then the body will sink into the heavy liquid, and the amount of displacement will be exactly equal to the quantity of the heavier liquid which will pass up into the interior of the body and above the level of the heavier liquid outside of the body.

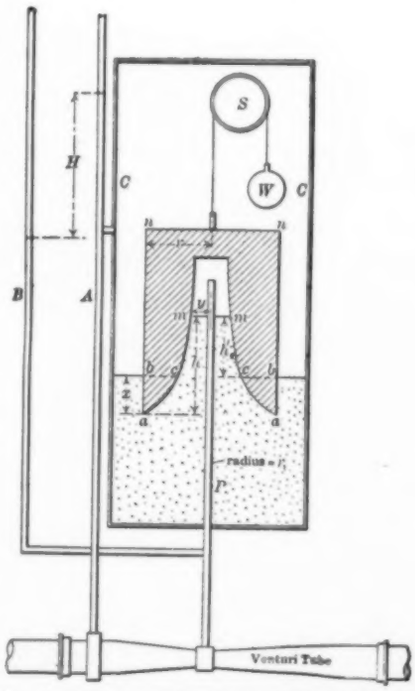


FIG. 1—METER REGISTERING THE FLOW OF LIQUIDS THROUGH VENTURI TUBE

Let the hollow body, or float, be represented by $n\ n\ a\ a$, Fig. 1; assume it to be a figure of revolution, and call the total volume of the heavy liquid, $a\ a\ m\ m$, within the float M.

The principle may be understood by a consideration of the laws of equilibrium—the algebraic sum of the vertical forces must be equal to zero; the sum of the horizontal forces, and the sum of the moments must all be equal to zero.

We need consider only the first condition as affecting the problem. At the plane $b\ c\ c\ b$ the heavier liquid is subject to the same unit pressure within and without the hollow body, but the resultant of external forces acting downward on the hollow body is measured by the weight of the heavier liquid, $m\ m\ c\ c$, and the total upward force acting on the body is the displacement of the annular portion of the floating body, $b\ c\ a\ b\ c\ a$.

For equilibrium, the algebraic sum of these two forces must be equal to zero, and, as one is positive and the other negative, they must be equal in amount; but, if that is the case, the entire quantity of liquid equivalent to the displacement of the hollow body, and no more, must be found in the interior of the hollow body, and hence the outer level of the surface of the heavy liquid does not change with variations of pressure outside and inside the float, as will be very easily understood.[1]

If there is a difference of pressure, H, then the float will sink into the heavier liquid an amount represented by the annular volume, $a\ c\ b$, which will be called V; and the net volume of liquid, $m\ m\ c\ c$, will also be equal to V, and h_2 will be proportional to H. As before stated, the level $b\ b$ will always be the same, no matter what the values of H, h_2, and V may be.

Let

x = the vertical movement of the float from a given datum;
H = the Venturi head;
h_2 = the corresponding mercury head;
v = the velocity through the Venturi tube;
y = the ordinate or variable radius of the inner portion of the float;
h = the corresponding abscissas, measured from the bottom of the float;
r = the constant radius of the outer cylindrical portion of the float;
M = the total volume, $a\ a\ m\ m$, of the heavy liquid within the float;
V_0 = the volume of heavy liquid above cc;
and
V_1 = the volume of the float below $bc\ cb$.

Assume the radius of the pipe, P, to be small and negligible. Take, by hypothesis, $v \propto x$, also for unity the coefficient, $x^2 = h_2$, for $v \propto \sqrt{H} \propto \sqrt{h_2}$.

Therefore, the cross-section of the float must conform to the following equation:[2]

$$y^2 = \frac{r^2}{2\sqrt{h + \frac{1}{4}}} \qquad (1)$$

This is an equation of the fifth degree, as between the two variables, h and y. No attempt has been made to discuss this equation exhaustively or plot all the loci, for, though this would be interesting from a purely mathematical standpoint, it is evident that only that locus will satisfy the physical conditions wherein both h and y have positive, real values.

For $h = 0$, $y = r$, and for $h = \infty$, $y = 0$, thus it is seen that there are no values of h between 0 and ∞ which give infinite or imaginary values of y. As a consequence, the integrating apparatus operated by the sheave, S, is theoretically accurate from zero up to any practical limit that may be desired.

The shape of the float is such that its vertical movement is proportional to the square root of h_2 or H (the mercury and water Venturi heads, respectively); and as the velocity through the Venturi tube is also proportional to the square root of H, the vertical movement is directly proportional to the velocity or quantity of water passing through the Venturi tube.

The same principle will hold true if the Venturi tube is replaced by any form of orifice, or if the pipes A and B be connected with Pitot tubes in a main; for, in any of these cases, the H values represent velocity heads, the same as they do for Venturi tubes, and the velocity heads are always proportional to the square of the velocity producing the head.

It is perfectly practical to have x less or greater than h_2, for, if we assume $cx^2 = h_2$, equation (1) becomes:

$$y^2 = \frac{r^2}{2\sqrt{c}\sqrt{h + c/4}} \qquad (2)$$

which is the more general form, in which we can give to c any value, consistent with practical limitations. For general use, however, equation (1) is satisfactory, particularly as there is no value of c which increases or diminishes the accuracy of the apparatus.

In these calculations, the radius of the pipe, P, within the float, is assumed to be so small that it does not affect the results. Even if no correction is made for this volume, the error at low velocities would be negligible, and, at maximum veloci-

[1] If the float were not free to move vertically, that is, if it were restricted in any way, as with a variable lever arm, counterweight or spring—then the movements of the liquids would not follow the law stated, but, as the difference of pressure became greater, the surface level of the heavy liquid would fall below a fixed level on the outside and rise above a fixed level on the inside.

[2] The proof of this equation which is given in the paper is here omitted.

ties, it would be less than half of 1 per cent, provided the pipe is made as small as possible. Its effect is to raise slightly the level of the heavy liquid, both inside and outside the float, by the same amount and aggregating the volume of displacement of the pipe P; and, where it is desired to make this inside pipe P of an appreciably large outside diameter, allowance must be made for its disturbing influence.

The final rationally derived formulas for that purpose are as follows:

$$x^2 = h_2$$

$$h = x + x^2 + \frac{r^2_1 x^2}{y^2 + R^2 - r^2_1 - r^2}$$

$$x = \frac{(r^2 - y^2)(y^2 + R^2 - r^2_1 - r^2)}{2y^2(y^2 + R^2 - r^2_1 - r^2) - 2r^2_1(r^2 - y^2)}$$

in which the only variables are x and y.

h_2 = the mercury rise as before;

x = the vertical movement of the float;

and

y = the variable radius of the interior.

The constants are:

R = radius of cylinder containing float;

r = radius of outer surface of float;

and

r_1 = radius of pipe P.

To use these formulas practically, it is necessary to tabulate x and y, h_2 and finally h; h representing the abscissas, as before, and y the ordinates. The necessity of this complication can be avoided by enlarging the outer cylinder at the location of the surface of the mercury.

Although the writer has devised many other forms and modifications, the form of float illustrated in Fig. 1, and covered by

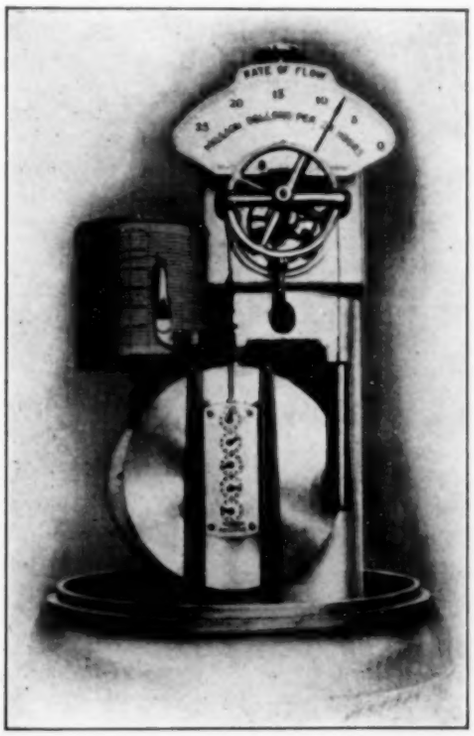


FIG. 2—TAP OF METER REGISTER WITH GLASS DOME REMOVED

equations (1) and (2), is believed to be by far the most practical, because it provides for maximum increments of volume and consequent accuracy at minimum velocities; and, with a float of a given radius, the minimum clearances are possible with the smallest cylindrical casing; also there are several other practical advantages of importance. . . .

One of the first difficulties to overcome was to prevent air from accumulating at the upper interior of the float in Fig. 1. Such an accumulation, of course, would lighten the float and therefore vitiate the results. Two check-valves opening upward are placed in the cap of the float. These are opened automatically when the velocity head reaches the maximum, and they can be opened at will, by a hand device, from the outside of the bottom of the pressure chamber, without otherwise dismantling the meter. This same device also automatically prevents the loss of mercury when the flow reaches an amount beyond the maximum capacity of the meter register, as it equalizes the pressure within and without the float.

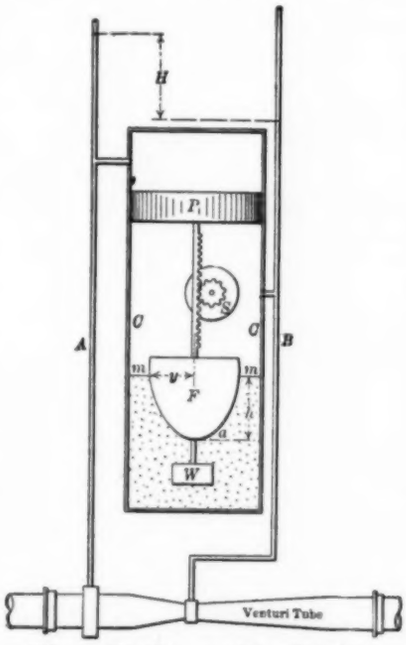


FIG. 3—ORIGINAL SIMPLE APPARATUS

An additional device for this purpose has been designed to prevent loss of mercury due to a sudden break of either of the pipes leading from the meter register to the Venturi tube, but these are safety devices of not unusual novelty, and, therefore, a detailed description of them is unnecessary.

Fig. 2 shows an application of this meter register.

The float is made of hard rubber to conform to a metal template; but, to make sure that it is correct and will perform according to the theory on which it is designed, each float is tested in a regular working machine by applying, under pressure, piezometric heads corresponding to the velocity heads of the Venturi tube or Pitot tube with which it is to be used, and noting from a standard rate-of-flow dial the corresponding velocities, which are compared with the theoretical velocities.

For heads, H, greater than 1/8 ft. the coefficient of velocity through a standard Venturi tube is so nearly constant as to secure results within 2 per cent of the actual. However, with the apparatus above described, Venturi heads, H, of less than 0.01 ft. are readily integrated. The velocity coefficient through the Venturi tube, however, is materially different, amounting to as much as 7 per cent at low velocities, and the instrument is modified accordingly so as to produce accurate results down to less than 0.01 ft.

The data for this modification, as well as the method of making the correction, are intended to be the subject of a later paper.

Leading up to the final development of the apparatus illustrated in Fig. 1, it will be interesting to consider two of the earlier forms. Fig. 3 shows an extremely simple apparatus, which, at first sight, would appear to be superior to the device shown in Fig. 1. P is a piston, F is a shaped float, S is a sheave, and W is a displacement device, all within a casing. The upper portion of this casing is connected with a pipe from the full section of the Venturi tube, and the portion below the piston P is connected with a pipe, leading to the throat of the Venturi tube. The differential pressure or Venturi head, H, acts on the piston and forces the float, F, down into the mercury.

As before, the problem is to shape the float so that its vertical movement from a fixed line of reference will be proportional to the square root of H. The displacement device, W, is such that when there is no flow in the Venturi tube and the apparatus is in equilibrium, the bottom of the float, a, will be tangent to the upper surface of the mercury. The level of the

mercury, $m\ m$, will change, of course, as the float descends. The equation of the float, though rather more difficult to deduce, is as follows:

$$y^3 = e - \frac{f}{\sqrt{h+g}}$$

e, f, and g being constants depending on the size of the piston, P, the assumed movement of the float, F, with the given Venturi head, H, and the relative weights of mercury and water.

The only radical difference in form between this formula and the one corresponding to Fig. 1 is in the constant e before the fraction. The curve, however, is still one of the fifth degree.

The objection to this apparatus is the friction of the piston P. This was recognized immediately, and, to overcome it, a diaphragm of an accordion or some very flexible type was substituted for the piston; this, however, introduced difficulties caused by the elasticity of the diaphragm and the necessity of modifying the shape of the float to provide for these variations, as the diaphragm could never be depended on to remain in a permanent position.

Another form is shown in Fig. 4. This is an entirely different principle, but the treatment of the problem is analogous to that of Fig. 1.

C represents a casing within which is contained a fixed displacing device F and a movable cylindrical vessel P containing, say, mercury buoyed up by the displacing volume W. The pipe leading from the full section of the Venturi tube connects as shown above the vessel, and the pipe leading from the throat of the Venturi tube connects within the displacing device F.

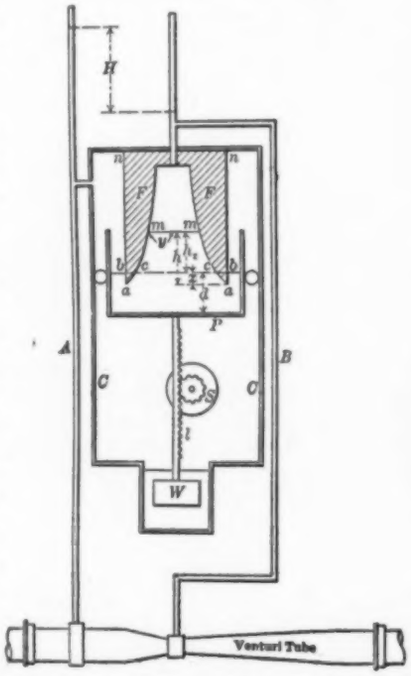


FIG. 4—ANOTHER FORM OF SIMPLE APPARATUS

The differential pressure, or Venturi head, H, forces mercury from the outside of the displacing device F to the inside; but for equilibrium, the depth of the mercury in the vessel P must always be the same; that is, d must always be constant.

With these data it is found that the equation representing the inner curve of the displacing device is exactly the same as that of the float in Fig. 1, namely, equation (1) given before.

This device, in some respects, is superior to any of the others considered, and, in fact, is was the one of which a complete working model was first made and tested, and the results were good. The only objection is the large quantity of mercury required for the displacing device, W.

It is obvious that the vertical movement of the cylindrical vessel, P, may be guided by rollers, as shown, or in some other practical manner. The cross-section of the rod, l, need not be sufficiently large to make its variable displacement affect the results appreciably.

However, the form in Fig. 1 is that which has been adopted as the most practical; and of all the twelve other devices, each of which has been developed by the writer to a working design, that shown in Fig. 4, next to that in Fig. 1, is the most meritorious.

In all cases the registering mechanism outside of the casing may be exactly the same as shown. It comprises a rate-of-flow dial, a chart device to record the rate of flow, and a total-flow dial. The first requires only a graduated dial and a hand actuated by the shaft of the sheave, S. The second requires a

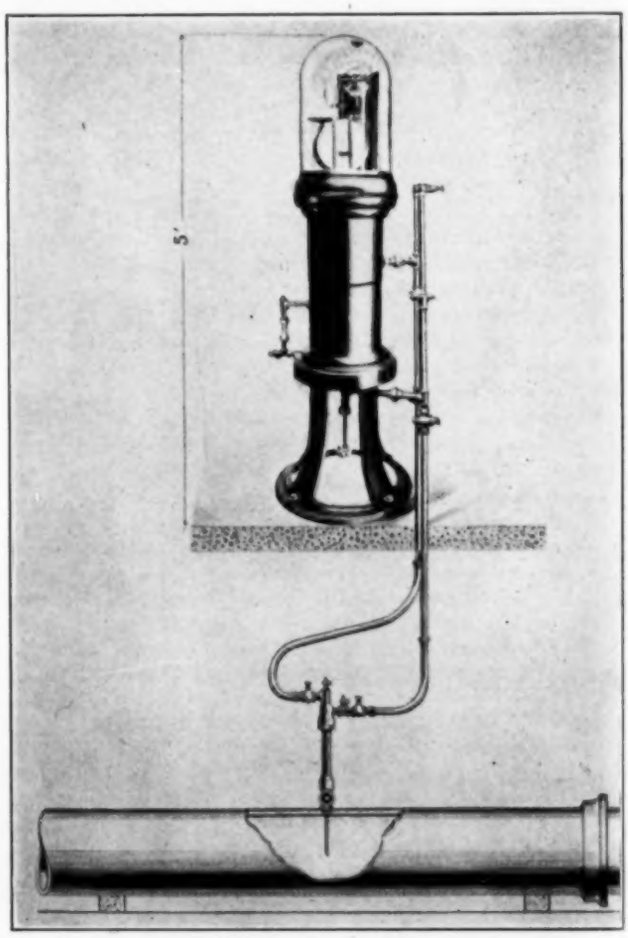


FIG. 5—STATIONARY METER WITH PITOT TUBE

pen reciprocating along the surface of a revolving figure having straight-line elements, such as a cylinder. The third requires a dial having hands which always revolve at a speed proportional to the angular position of the sheave, S.

This is readily accomplished by hanging from the sheave, S,

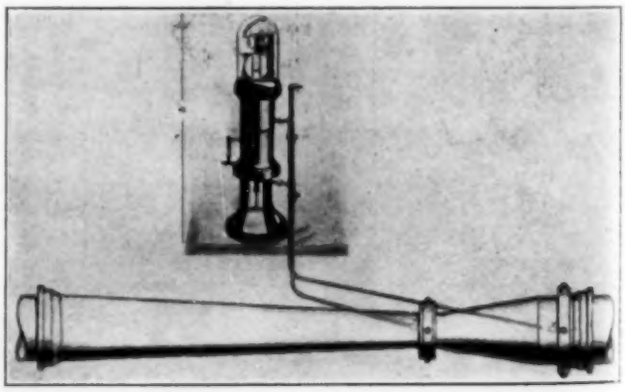


FIG. 6—STATIONARY METER WITH VENTURI TUBE

the dial which is connected with and operated by a traction wheel, the latter being rotated by a constantly revolving disk. When the sheave, S, sets at zero, the traction wheel stands at the center of the disk, and, therefore, has no rotating motion. As the flow increases from zero and the sheave, S, revolves

the traction wheel moves away from the center of the disk and therefore revolves at a speed proportional to its resistance from the center of the disk and to the rate of flow through the Venturi tube.

There are other equally simple means of actuating the total-flow dial, but they are all on the same principle, and have been in use for several decades. The revolving flat disk is probably the oldest, but the revolving cone rivals it in antiquity. The Nicholson ship's log is a rather recent application of the latter device. It will be seen from Fig. 2 that the chart-recording cylinder and the disk are operated by the same clockwork.

Where it is intended to meter a liquid of a specific gravity different from that for which this meter is adjusted, as, for

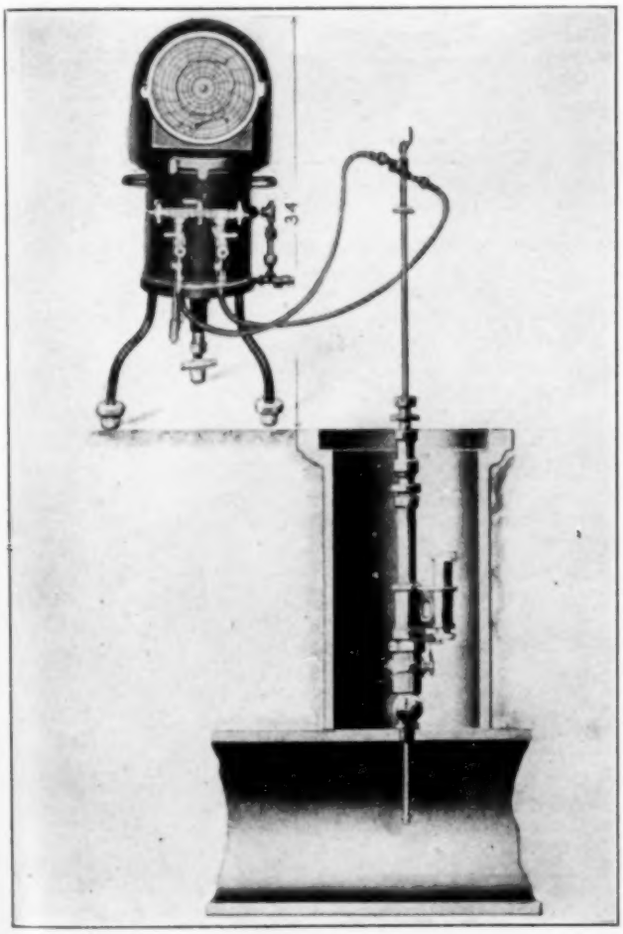


FIG. 7—PORTABLE PITOT RECORDER FOR MEASURING AND RECORDING FLOW OR LEAKAGE OF WATER MAINS, FLUMES, PENSTOCKS, ETC.

instance, if the meter has been adjusted for measuring water at 50 deg. Fahr., and it is desired to measure water having a temperature of 200 deg. Fahr., the results can be corrected according to the following principle:

In a fluid passing through a Venturi tube or orifice, the weight is directly, and the volume is inversely, proportional to the square root of the unit weight of that fluid, for:

Letting

W = the weight of fluid,

Q = the volume of the fluid,

H = the Venturi head,

and

w = the unit weight of fluid;

then

$$Q \propto \sqrt{H}$$

$$H \propto \frac{1}{w}$$

$$W \propto wQ$$

From these equations we obtain

$$Q \propto \frac{1}{\sqrt{w}}$$

and

$$W \propto \sqrt{w}$$

The latter two equations are the stated relations.

TABLE I.—TEST OF A METER REGISTER BASED ON VENTURI HEADS. FLOAT NO. 46.

C	V	V_1	D	P
0.010	0.195	0.194	+0.001	+0.52
0.013	0.226	0.225	+0.001	+0.45
0.023	0.300	0.298	+0.002	+0.67
0.036	0.381	0.379	+0.002	+0.53
0.054	0.470	0.467	+0.003	+0.64
0.078	0.571	0.562	+0.009	+1.60
0.112	0.677	0.675	+0.002	+0.30
0.154	0.800	0.797	+0.003	+0.38
0.208	0.931	0.930	+0.001	+0.11
0.279	1.079	1.080	—0.001	—0.09
0.368	1.240	1.240	0.000	0.00
0.487	1.429	1.428	0.001	0.00
0.638	1.635	1.635	0.000	0.00
0.833	1.872	1.874	—0.002	—0.11
1.092	2.150	2.133	+0.007	+0.33
1.426	2.460	2.460	0.000	0.00
1.873	2.820	2.825	—0.005	—0.18
2.455	3.250	3.235	+0.015	+0.47
3.250	3.750	3.720	+0.030	+0.81
4.320	4.321	4.280	+0.041	+0.96
4.830	4.570	4.540	+0.030	+0.66
5.800	5.010	4.970	+0.040	+0.81
7.270	5.638	5.600	+0.038	+0.68

Table 1 contains data relating to a typical test[8] of meter registers constructed on the principle described herein, in which

C = the Venturi head as measured, in feet;

V = the theoretical velocity through the full section of the Venturi tube, in feet per second;

V_1 = the velocity as indicated by the meter register;

D = the differences;

and

P = the percentages of error.

This meter register is in use in several important cases for recording and integrating the flow measured by Pitot tubes, and has also recently been adapted in portable form utilizing Pitot tubes for city main leakage survey purposes. This form is shown in Fig. 7. Quite a number of these instruments are already in service. (A typical test of one of these recorders is given in the paper, but not reproduced here.)

The Carnegie Institute of Technology has issued an instructive pamphlet entitled "Suggestions Concerning a Course in Engineering." Adaptability for engineering work consists in the possession of powers of observation and reasoning; interest in the discovery and application of nature's laws; a strong imagination, ingenuity and resourcefulness; keenness to see the application of mathematical principles to mechanical problems; and practical common sense combined with habits of care and accuracy. In addition to outlining the requirements for engineering in general, the pamphlet then gives the specific requirements for chemical, civil, mechanical, electrical, commercial, mtallurgical, mining and sanitary engineering, and states the facilities offered for study in these courses at the Carnegie Institute of Technology, Pittsburgh.

Indian pig-iron is being imported into the United States in small quantities, as an experiment to see whether Bombay producers can compete in this market. About 600 tons has been sent to San Francisco, and more will follow if the iron is sold at satisfactory prices. The Indian iron is said to be of superior quality, and it is on this that the producers are relying to create a demand for their product. The cost of production in India is quite low, but the transportation and duty are high.

[8]The paper contains in a table the results of a second test which are not reproduced here.

Evolution of Methods of Handling Slime. IV.

THE RAND, SOUTH AFRICA

By H. N. Spicer

The marvelous development of the resources of the Transvaal dates from about 1883, at which time the first stamp mill was erected to treat gold ore from the De Kaap goldfield. Many interesting comparisons could be made between conditions existing then and now, showing the gradual development of metallurgical methods and the increasing magnitude of the industry. It will suffice, however, to give the reader an impression of the wonderful development on the Rand by showing in Figs. 1 and 2 views of the first 10-stamp mill erected in 1883, and one of the modern 600-stamp mills built in 1911. Comparison of the two will give an idea of the large scale of present operations; and this can be reinforced by statistics which show that in 1912 there were over 10,000 stamps in daily operation, and that the value of gold produced in that year was nearly $200,000,000, recovered from about 25,500,000 tons of ore.

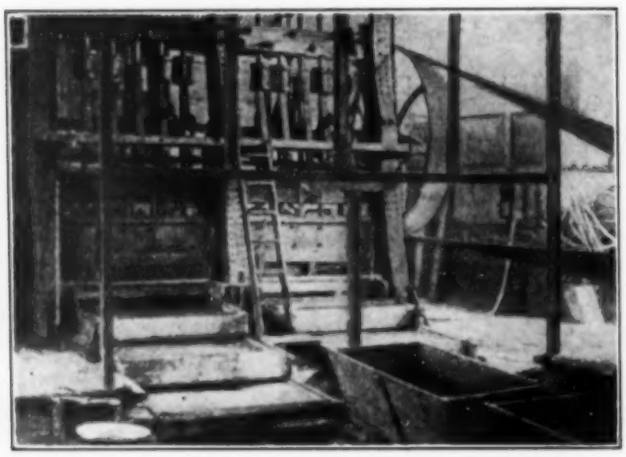


FIG. 1.—BROWN'S HILL 10-STAMP MILL, ERECTED IN 1883

First Slime Treatment in 1894

The first cyanide plant on the Rand was built in 1890 at the Salisbury mine, but at that time and for several years thereafter only sand was treated by leaching. It was not until 1894 that any effectual attempt was made to treat slime. In that year Mr. W. Bettel did some interesting work at the Primrose mine, using an excess of lime to accelerate slime settlement. His experiments demonstrated the value of this treatment, and probably initiated an important step in cyanidation which made slime treatment a commercial success on the Rand. In this connection the views shown in Figs. 3 and 4 are of historic interest. Fig. 3 shows a bird's-eye of the first slime treatment plant, at the Primrose; and Fig. 4 a closer view of a settling basin in which the slime pulp was mixed with lime and allowed to settle. The supernatant clear water was decanted through a submerged launder, and a pump was used to remove the balance of the water remaining between the level of the launder and the surface of the slime. The operation was crude compared with modern methods of continuous thickening and decantation, but it shows from what insignificant beginnings the modern method of slime treatment has developed.

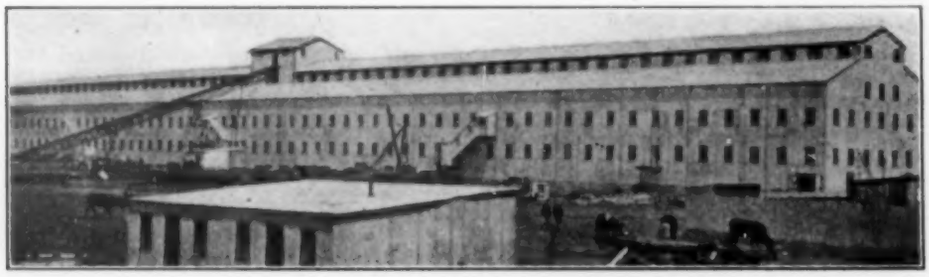


FIG. 2.—RANDFONTEIN CENTRAL 600-STAMP MILL

FIG. 3.—BIRDSEYE VIEW OF FIRST EXPERIMENTAL SLIME PLANT, PRIMROSE MINE, 1894

The practical application of the experimental work done by Mr. Bettel fell to the lot of Messrs. Williams and Butters, who initiated the intermittent decantation process which today is used in practically all the mills of the Rand.

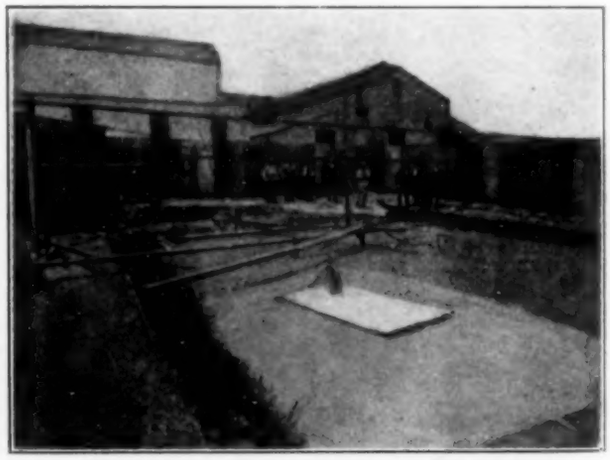


FIG. 4.—SLIME SETTLING BASIN AT PRIMROSE MILL, SETTLEMENT WITH LIME

Effect of Local Influences

For many years during the early history of the Rand, the development of the cyanide process was largely in the hands of local metallurgists, and was affected but little by the practice in other countries. And while it is true that the metallurgy of gold ores has been conducted there on a scale not equalled in magnitude elsewhere, nevertheless it has not kept pace, on the whole, in the adoption of methods and machines that are in daily use in other countries, and this is particularly true with respect to the handling and the treatment of slime.

During the last year or two some very marked changes have taken place in Rand metallurgical practice, and there is little doubt that more modern methods will be adopted in the

mills built in the future. The successful features of cyanidation as practiced in other countries are gradually finding acceptance, and will replace the less efficient local methods.

FIG. 5.—TUBE MILLS AT THE BRAKPAN MILL

Crushing in Water is General Practice

With one or two exceptions it is common practice on the Rand to perform the initial crushing and grinding in water. This entails the subsequent removal of water from both sand

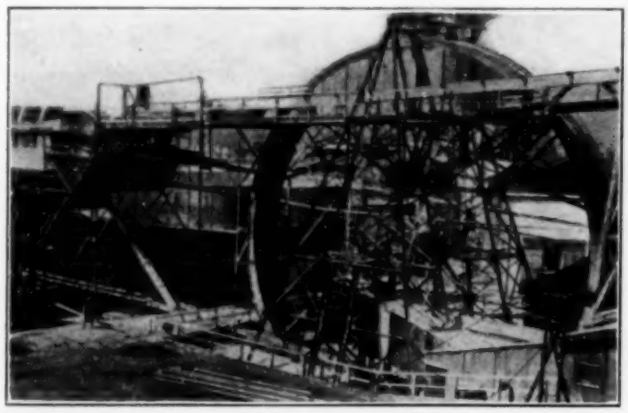


FIG. 6.—TAILING WHEEL ELEVATING PULP TO CLASSIFIERS AT NEW GOCH

and slime before cyanide solution can be applied, and introduces a step which is difficult of satisfactory accomplishment, particularly in treating slime by intermittent decantation. The first attempt at crushing in cyanide solution was made at the New Goch in 1906, but for various reasons it was abandoned.

Heavy gravity stamps and tube mills are used as crushing machines throughout the Rand, and some very large installations of this type have been made. Formerly, as elsewhere, amalgamation followed stamp milling; but with the introduction of the tube mill at the Glen Deep in 1904, and its subequent general adoption, the practice was changed, and today it is customary to amalgamate after tube milling. A marked change also has been made in the area of amalgamation surface used; and even in mills of recent construction, such as the Randfontein Central, more than half the amalgamation area is not in service. Rand tube mill practice has become standardized, and the customary type of tube is 22 ft. long and 5 ft. 6 in. in diameter. Six of these are shown in a view of the unfinished Brakpan mill, Fig. 5.

A common method of elevating the mixed sand and slime pulp from the grinding machines to the classifiers is by the well known tailing wheel. The latest design is similar to a huge bicycle wheel, as shown in Fig. 6.

Cone Classification of Sand and Slime

Cones are generally used as sand-slime classifiers. From the view given in Fig. 7, which shows the system in use at the City & Suburban, the reader will gain an idea of the size of the cones. The system has its disadvantages in the necessary loss of head, and in the double treatment which must be adopted in order to remove the last portion of fine sand from the slime.

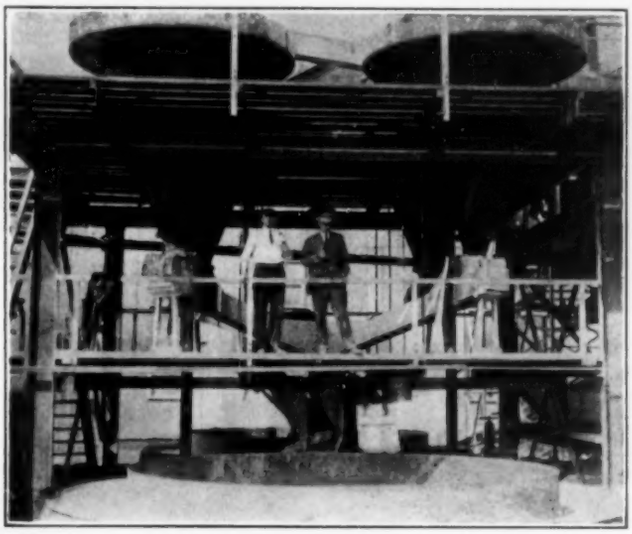


FIG. 7.—CONE CLASSIFIERS AND SAND FILTER TABLES AT THE CITY AND SUBURBAN

Rotary, vacuum filter-tables of local invention are used to dewater the sand. As shown in Fig. 7, they are placed below the cones and receive their underflow. These machines have a peripheral filtering surface about 30 in. wide, constructed in the usual manner and connected with a suitable vacuum pump. They do excellent work, reducing the moisture in the sand to an average of 13 per cent. The cost of operation, however, is considerable, as a 20-ft. table consumes about 10 hp for vacuum and rotation. The cost of maintenance also is high, as the filter cloth has to be removed and cleaned once in 24 hours, due to the fact that the classified sand still contains enough slime to clog the pores of the filter. As a consequence it is necessary to have at least two tables in any installation.

FIG. 8.—SAND LEACHING VATS AT RANDFONTEIN CENTRAL

The necessity for obtaining such a low percentage of mois-

ture in the sand is a consequence of crushing in water. If the ore were originally crushed in cyanide solution, the necessity would not exist, and the filter-tables could be dispensed with.

The dewatered sand is mixed with cyanide solution and either pumped or raised by a wheel to collecting tanks, from

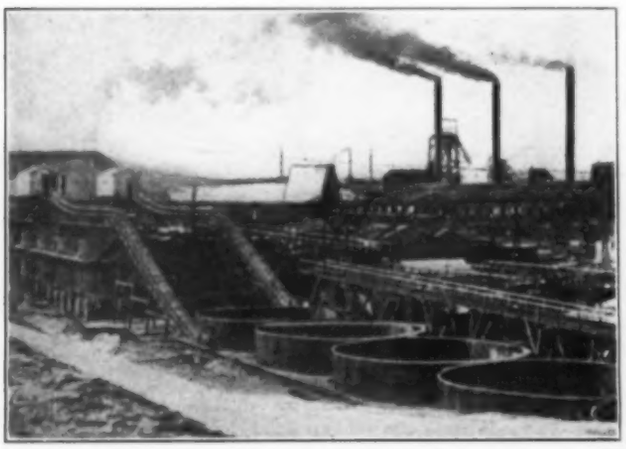


FIG. 9.—SAND LEACHING VATS AT ROBINSON DEEP, WITH BELT CONVEYORS

which the solution overflows through gates. It is customary to give the sand a double treatment: either in superimposed sets of vats, as shown in Fig. 8, or in separate sets as in Fig. 9. In the first case the sand is shoveled from the upper to the lower vats after the first treatment; in the second it is transferred by means of belt conveyors, as shown in the figure. With but few exceptions the collectors and leaching vats are unloaded by hand.

Variable Quantity of Slime Produced

The quantity of slime produced in the fine crushing of Rand ores depends largely on the depth at which the ore is mined. Ore from the outcrop or shallow mines will produce as much as 35 per cent true slime, whereas the hard ore from the deep mines produces none at all, the so-called slime being only very fine sand. Pyrite is found in some of the deep-level ores in quantities as high as 2½ per cent, but it offers no obstacle to extraction if it is ground very fine. Owing to

FIG. 10.—SLIME VATS AT VILLAGE DEEP

the wide difference in the classes of ore, there is considerable variation in the rate of slime settlement at the different mills.

Until within the last two years there had been but little real change in the method of treating slime since the day when Butters and Williams introduced intermittent decantation. This system offers great disadvantages and is perhaps the weakest point in Rand metallurgical practice. Owing to the fact that the ore is crushed in water, the necessity arises for preliminary thickening by settlement, and the decantation of the water before cyanide solution can be added. In order to aid settlement as much as possible, and at the same time neu-

FIG. 11.—ROBINSON CYANIDE PLANT FOR DECANTATION OF SLIME

tralize the acidity of the ore, lime is added at the stamp mills. It is imperative that the slime be dewatered to a minimum percentage of moisture, as otherwise large quantities of cyanide solution would have to be wasted continually.

Intermittent Decantation

The cyanidation of slime by intermittent decantation requires large areas of collecting and settling tanks. These are characteristic of all Rand cyanide mills, and form a large part of every plant. Typical views of such installations are shown in Figs. 10 and 11.

The size of slime tanks usually found on the Rand is from 50 ft. to 70 ft. diameter and 12 ft. to 15 ft. deep. The bottom is slightly conical to facilitate discharge of the settled slime.

The diluted slime pulp overflowing from the cone classifiers enters the collecting tanks at one or several points near the center, flowing into wells which restrict undue agitation of

FIG. 12.—SLIME TREATMENT VATS AT THE BRAKPAN MINES

the charge and prevent disturbance in the settling area. The inflow is so regulated that a perfectly clear overflow is secured at the periphery of the tank. When the overflow will no longer run clear, the feed is diverted to another collector, and settlement is allowed to proceed. The time allowed for this operation depends on the nature of the slime and varies

between wide limits. As soon as settlement is complete the supernatant water is decanted through adjustable pipes.

In the foreground of Fig. 12 are shown some slime collectors arranged for loading through central wells. The rear tank at the left is being filled, and clear water is overflowing at the periphery. The corresponding tank at the right is

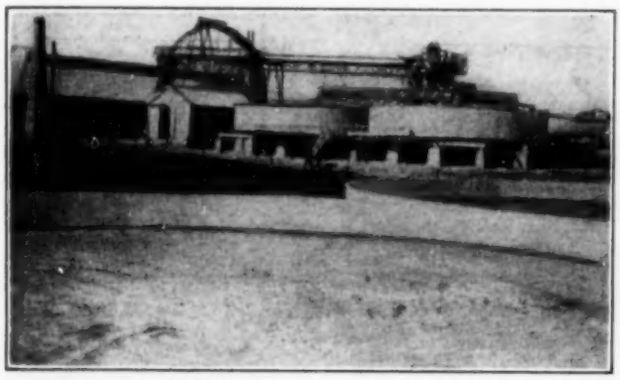


FIG. 13.—AGITATION IN SLIME VATS AT ROODEPORT UNITED

being decanted. Incidentally, the view shows one of the methods adopted for stacking sand tailing.

As mentioned before, the rate of settlement and the density of the settled slime vary widely with the nature of the ore. Speaking broadly, it is found that slime produced from the outcrop or shallow mines will settle to about 50 per cent moisture, while that from deep mines will settle to 42 per cent or 45 per cent moisture. At some of the plants advantage is taken of the fact that the rate of settlement increases with the temperature, and various methods are adopted to heat the pulp.

Following decantation of the water, the settled slime is sluiced from the collector with cyanide solution, and transferred by centrifugal pumps to the first treatment vat in which the operation of settling and decantation is repeated. The strength of solution varies from 0.0005 per cent to 0.02 per cent KCN, and the ratio of solution to solids is 4 or 5 to 1.

Agitation Not Generally Practiced

As a whole, the slime from Rand ores requires but little agitation, and until recently no special provision has been made for that operation. Dissolution of the gold occurs during the

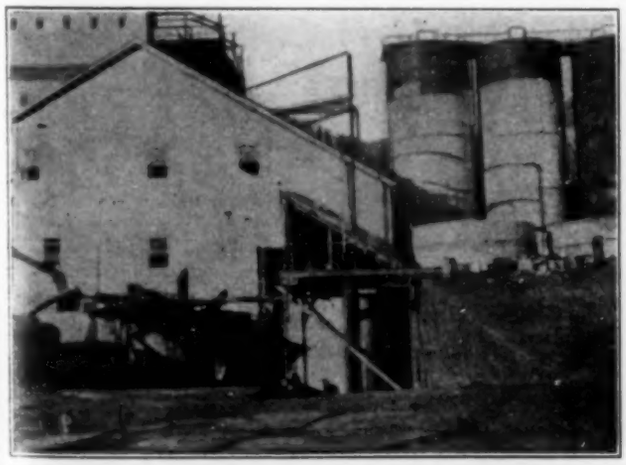


FIG. 14.—PACHUCA AGITATORS AT FRENCH BOB'S

transfer of pulp from the collectors to the treatment vats. This, however, is frequently insufficient to dissolve the maximum amount of gold, and is commonly supplemented by circulating the charge in the vat. This operation is shown in Fig. 13, which shows the swirling motion imparted to the contents of the vat.

The time available for this operation seldom exceeds three or four hours, as the vat is needed for the next lot of collected slime. In addition to the fact that the time is insufficient to accomplish the purpose, the shape of the vat is ill adapted to keeping slime in suspension, and thereby effectually aid dissolution of the gold. Realizing the deficiency of the method, and the advantage to be gained in some cases by genuine agitation, some companies have adopted the Brown or Pachuca agitator, as shown in Fig. 14.

Returning to the decantation process: After the charge is agitated, or rather mixed, the slime is allowed to settle and the clear solution decanted. The thick slime is then sluiced from the vat with barren solution, and again transferred by centrifugal pumps to a second treatment vat. But little circulation is required in this vat, as most of the gold has been dissolved and removed; and when the transfer and mixing is complete, settlement and decantation proceed as before. The only solution precipitated is that decanted from the first treatment. Ordinarily this is clarified by passing through sand filters, but lately plate and frame presses have been adopted for this purpose.

Usually this ends the treatment of the slime, but in some instances a third dilution is required to remove the last portion of dissolved gold. In that event the settled slime is again transferred with solution to a third vat where the same operation is repeated. Finally the impoverished slime is sluiced

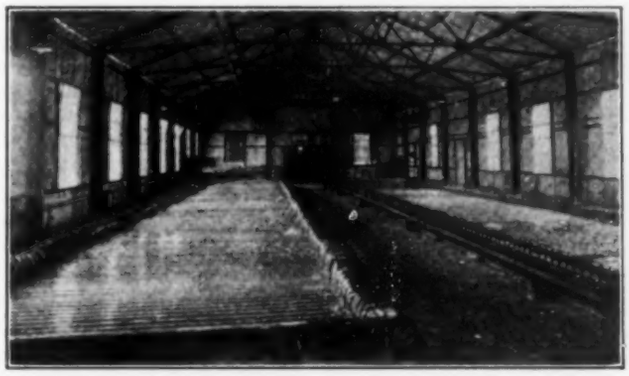


FIG. 15.—BUTTERS' VACUUM SLIME FILTER AT THE BRAKPAN MINES

from the vat with the least water possible, and impounded in dams from which considerable water is recovered and pumped back to the mill. The solutions decanted from the second and third dilutions are variously used as wash solutions in sand treatment.

It will not be surprising to the reader to learn that attempts have been made from time to time to supplant this cumbersome method of slime treatment. As far back as 1903 filter pressing was introduced for this purpose. Continuous decantation in cones also was tried, with the idea of reducing the value of the slime before filter pressing. These attempts proved abortive, however, largely on account of the mechanical difficulties encountered in the use of cones, for at that time no better device than the cone was known for continuous decantation. Had the modern continuous thickener been known then, it is possible that the whole history of slime treatment on the Rand would have been changed. Some of these modern machines have been used recently and probably will be more extensively adopted.

Vacuum Filtration in Newer Mills

It was not until 1910 that the first vacuum slime filter was installed on the Rand, despite the fact that the system had long been in successful operation in other countries. The first installation aroused some pessimistic and skeptical comment, but the success of the process was such that today it is the greatest rival of intermittent decantation on the Rand. Eight of these plants are now in operation, treating an aggregate of 6500 tons of slime per day, and there is every indication that

the process has become an integral part of Rand slime treatment. Fig. 15 shows a view of the Butters filter at the Brakpan mines, handling 1000 tons of slime per day.

Notes on Chemistry and Metallurgy in Great Britain

(From Our Special Correspondent)

Iron and Steel Institute

The annual meeting of the Iron and Steel Institute was held on May 1st and 2nd at the Institute of Mechanical Engineers, under the presidency of Mr. A. Cooper.

The Bessemer Medal was awarded to M. Adolphe Greiner.

The first paper read was on "The Production of Sound Steel by Lateral Compression of the Ingot while its Centre is Liquid." The author, Mr. **Benjamin Talbot,** advocates the advantage of using aluminium and similar deoxidizers to reduce the amount of segregation and prevent blow-holes in the outer portions of the ingots; and he also employs a process of lateral compression after the ingots have been in the soaking pit for a definite time and before central solidification occurs.

The discussion was opened by Sir Robert Hadfield, who asked if the author had taken the casting temperature into account, what would be the effect of a greater percentage of phosphorus and if traverse tests and tests to destruction had been taken.

M. Adolphe Greiner remarked that whereas the author indicated that the object of his paper was to give a comparison between his own method and the result of the present-day practice, he did not really give any such comparison, and he therefore asked him to do so.

Mr. H. Savage referred to trials made at Seraing, the results of which indicated inferior strength of rails produced from laterally compressed steel as compared with those from ordinary metal.

Mr. R. T. Scott remarked that sections of a very large number of ingots compressed by the Harnet process, when sliced down the middle and planed to the exact centre, did not show any line of demarcation or segregation like those which Mr. Talbot had found in his ingots.

Mr. F. W. Harbord said he had examined specimens of rails and ingots from steel made by the author's process, both with and without aluminium, and it seemed that it was necessary in order to get a sound outside to have some deoxidizing agent. The difficulties from the practical point of view were very great, and it remained to be seen how far they could be overcome in every-day practice. It appeared to him that there would be always a danger that if the ingots were squeezed too soon they would bleed, or that the segregate would squeeze out too near the surface; and, on the other hand, if the squeezing were done too late the piping would not be closed up.

Mr. J. M. Gledhill said that his firm (Sir W. G. Armstrong, Whitworth & Co.) had continued to carry out compression in all their ingots ever since the introduction of compressed steel. He believed that Mr. Talbot was on the right track, but the difference of casting temperature gave rise to the greatest possible trouble. He had always found that one of the greatest advantages of compressed steel ingots was the entire absence of blow-holes on the surface.

Mr. F. W. Paul said that the bleeding effect seemd to him to be one of the most serious difficulties in practically carrying out the new process. He had frequently analyzed metal which exuded out; and even with a soft ingot, containing an average of 0.17 carbon, segregation had taken place up to as much as 0.6 carbon, and the portion which had bled out was the highly segregated portion. Another important matter which was mentioned by Mr. Talbot was that the process depended on a time-table, and this, he thought, would prove the crux of the whole question; because the physical properties of molten steel varied very much, and what was known as a lively steel took much longer to cool than a steel which was termed dead.

Mr. E. H. Saniter said that the liberation of nitrogen by aluminium, mentioned in the paper, appeared to him to be a novel idea, and it seemed to him that if aluminium disengaged gas from the steel it was likely to make blow-holes rather than solid metal.

Mr. C. J. Bagley said it was well known that when pigs were broken from the sow some liquid metal escaped which was always found to contain more phosphorus and sulphur than any other part of the pig or sow. This showed that the segregate was squeezed out from the centre, which was the last part to solidify. This was the worst position for the segregates, but by Mr. Talbot's process they were taken from the centre to a position between the centre and the outside, where they were harmless.

The president remarked that the author spoke of experiments made with a 25-inch ingot. That was rather a large size, and there were not many mills in this country which could deal with it. He would like to ask Mr. Talbot whether he had arrived at any conclusion as to how much smaller ingots could be made for use with his process. It was clear that an ingot of 10 or 12 inches could not be successfully worked, because it would not be possible to maintain the temperature requisite to roll out the piping.

Mr. Benjamin Talbot, in his reply, said that with regard to the question of casting temperature he had found the use of a pyrometer unsatisfactory, and the temperature he employed was the practical temperature at which a clean ingot was cast. With regard to high phosphorus steel he presumed that reference was intended more especially to phosphorus in American steel, which for the last year or two was for Bessemer steel 0.10; but they had only to look at the present records of the American rail trade to see that it was reverting as quickly as it could to the basic open-hearth process. This process could not be adapted to every kind of ingot. There must be uniform conditions and, therefore, necessarily a certain size, and he had started on a size of ingot which worked successfully.

The next paper taken was by Mr. **Andrew Lamberton** on "A New Form of Electrically-driven two-high Continuous-running Reversing Mill," but the discussion did not afford any points of more than ordinary interest.

Mr. **P. Longmuir** next read a paper entitled "Studies in the Cold Flow of Steel," in which he also dealt more particularly with the effects of silicon, aluminium, and high carbon, and said that he found that the effect of cold working on the tensile strength of steels ranging between 0.1 and 0.89 per cent of carbon does not show any great variation. A sound uniform hot-rolled section is necessary for a successful flow; and segregated cores may give rise to cupping, which may also follow from excessive reduction at any one drawing. The discussion, in which Drs. Rogers and Rosenhain, Mr. Saniter and Sir Robert Hadfield took part, dealt with the question of stresses in drawing and its effects on crystalline structures.

Professor **J. von Ehrenwerth** then read a paper on "The Economy of Dry Blast," which, he contended, saved fuel, caused less strain on the blowing engines, produced a higher temperature in the melting zone, with more regular working of the furnace. In the discussion Mr. Greville Jones agreed that the dry blast was advantageous for ferro-silicon and ferro-maganese.

Mr. A. K. Reese said that each particular furnace had its own maximum temperature of best work, and moisture in the blast caused fluctuations in the melting zone which would not be caused by the dry blast.

Mr. Hutchinson did not think that dry blast would be economical in the Cleveland district, where they had a fairly dry atmosphere.

The next paper read was by Mr. **W. H. Hatfield** on the "Influence of Sulphur on the Stability of Iron Carbide in the Presence of Silicon." The author concludes that sulphur augments the stability of iron carbide at high temperatures.

possibly in consequence of a small quantity of the sulphur remaining associated with the carbide crystals, but the action is of a chemical nature; and Levy's suggestion of mechanical action by sulphide films is not supported; the effect of sulphur is largely neutralized by silicon and by manganese.

In the discussion Dr. Rogers suggested that the addition of sulphur in certain other forms would perhaps have resulted in the melting point of such additions not being reached as it had been in the author's experiments.

Dr. Rosenhain, Mr. H. I. Coe and Mr. Hatfield also took part in the discussion, which concluded the first day's papers.

SECOND DAY

The proceedings of the second day began with the presentation of the Andrew Carnegie gold medal to Dr. J. Newton Friend.

The first paper was on "Some Fundamental Faults of the Present Day Furnaces and Their Remedies," by Mr. **Alleyne Reynolds**, who advocated the production of the reducing atmosphere in the Siemens steel furnace and the utilization of the heat of the waste gases for heating the air. He claimed that a saving of about 25 per cent of fuel would be effected.

Mr. A. D. Ellis, who opened the discussion, said that the author's furnace reduced the quantity of scale and slagging and would be most useful for heating high carbon steels and for melting bronzes.

The next paper was on "The Tenacity, Deformation and Fractures of Soft Steel at High Temperatures," by Dr. W. Rosenhain and Mr. J. C. W. Humfrey who came to the conclusion that the behavior of such steel at high temperatures resembles that of an aggregate of crystals contained in a viscous fluid; while when cooled below Ar3 the crystals are weakened and yield before the cementite does, and consequently the strength of the material is then dependent on the size of the crystal increasing as the quantity of cement diminishes. On the allotropic theory of the hardening of steel the authors only say that one important transformation takes place in iron at about 900° C. and the lesser change at the temperature of Ar2, and in view of their results they are unable to support Benedicks' theory of the nature of beta iron.

The discussion, which was of a most spirited nature, was opened by Prof. J. O. Arnold, who said that that they were actually assembled to bury that poor, battered, threadbare old myth hard beta iron, and by the irony of fate the gravediggers were named Rosenhain and Humfrey, while the officiating clergy were Drs. Carpenter and Stead. They had before them two papers: one of comparatively small importance by Dr. Carpenter, and one of profound importance by Dr. Rosenhain and Mr. Humfrey.

With regard to Dr. Carpenter's paper, he would suggest that the memoir should have appeared under the joint names of Drs. Carpenter and Stead: the voice was Henry's but the hand was the hand of John. Dr. Carpenter's paper might be summed up in saying that in his almost pure iron the absorption curve presented only the point of Ac3, while the recalescence curve presented both the points Ar3 and Ar2. But Dr. Carpenter asserted that the latter was only the end of the former because some groups of gamma molecules had been rendered metastable by investing membranes 2 to 3 molecules thick, resulting from traces of impurities. He, in accordance with Dr. Benedicks, said that at Ar2 the metastable gamma molecules became unstable and changed into the alpha molecules with a slight evolution of heat: consequently there was no such thing as beta iron, but only alpha and gamma. Unfortunately Dr. Carpenter's theory was as thin as his hypothetical membranes, and the application of a small bevelled straight edge showed that Ac2 extended along a considerable amplitude in all Dr. Carpenter's inverse rate heating curves.

Professor Arnold proceeded to find fault with Dr. Carpenter's statement that the analysis made by Professors Hicks and O'Shea of the pure electrolytic iron had never been published. That statement was quite unfounded. It was published in the commemoration volume of the University College of Sheffield, to which the speaker referred in his British Association lecture in 1910, and also appeared in the discussion on Dr. Benedicks' paper before the Iron and Steel Institute in 1912, to both of which Dr. Carpenter had made reference. After further technical details Professor Arnold said it was perhaps fortunate for Dr. Carpenter that his paper had fallen to the ground, as otherwise he would have to assert that the magnetic change point at Ac2, or about 760° C., was really at Ac3, about 910° C. Equally, he would have laid to rest that mythical constituent martensite, which was supposed to be a highly crystalline solid solution of iron and carbide in beta iron. It was obvious that he could not possibly have a solid solution of carbide of iron in an allotropic modification of iron which, according to him, did not exist. He would also have to admit that 13 per cent of carbide of iron was soluble in the alpha range of temperature and that such solid solution was a micrographically amorphous and quartz-hard substance called hardenite, discovered by Sorby a lifetime ago.

Having thus mildly admonished Dr. Carpenter, Professor Arnold turned his artillery on Dr. Rosenhain and Mr. Humfrey. It would not be possible to do justice to Professor Arnold's criticisms if they were condensed, and your correspondent must therefore rest content for the present with saying that point by point he steadily and mercilessly demolished the authors' arguments and rendered their position untenable.

Engineering Imports and Exports

The returns issued by the Board of Trade for the four months ended April 30th continue to show very satisfactory features. As compared with the corresponding period of last year the imports of iron and steel, including manufactures, showed an increase of £1,552,962, and totalled £5,236,759; while exports amounted to £18,341,167, showing an increase of £3,333,944. In other metals, including manufactures, the imports touched £11,026,974, an increase of £1,117,153, and exports £4,532,727, an increase of £903,070. The imports of electrical good totalled £506,145, and showed a decrease of £24,667; exports went up to £1,916,285, a rise of £473,008. In machinery imports were £2,512,131, an increase of £297,593, and exports expanded to £11,934,657 and increased by £1,564,421. Imports of new ships with a total of £11,508 showed a decrease of £89, while exports put at £2,383,054 increased £666,121.

Taking the month of April alone, imports of iron and steel increased £495,069 and the exports went up by £2,351,651. Other metals increased £569,403 in imports, and improved £327,219 in exports. Imports of electrical goods were £12,255 higher and exports were £123,490 better. The imports of machinery increased £96,184 and the exports by £786,828. Imports of ships went up £7,878, while exports decreased £197,640.

Market Prices

May, 1913

Tin opened high £232, dropped to £230 in a few days, rose on the 7th to £231½, and dropped away from this point to £221 by the 15th. Then recovered to £224 (16th) and has since, after a drop to £218, kept about the £221 level, closing lower at £212.10.

Copper has been rather unsteady, rising from £67.5 to £69.5 in the first week, appreciating another 15/- in the second, dropping a pound in the third, then recovering a little, and closing £66.15.

Haematite has been steady throughout the month, but suffers slightly at the close 77/9.

Scotch Pig has declined in the same way as Cleveland, opening at 73/3, it closes at 66/6.

Cleveland remained steady at 67/9 till mid-month, then dropped to 68/6, recovered about 20th to 70/-, but since declined and sharing in the general depression closes at 60/6.

Lead has made a very steady rise throughout the month, opening £18.10 and closing £20.5.0.

	£ s. d.
Aluminium, per ton	90. 0. 0.

	£ s. d.
Alum, lump, loose, per ton	6. 0. 0
Antimony, black sulphide powder, per ton	20. 0. 0
Borax (crystal), per ton	17.10. 0
Copper ore, 10 to 28%, per unit	12/4½ to 12.10½
Copper sulphate, per ton	22.15. 0
Carbolic acid, liquid, per gal	1. 4
Caustic soda, 70%, per ton	9.12. 6
Ebonite rod, per lb	4. 6
Hydrochloric acid, cwt	5. 0
India rubber, Para, fine, lb	3. 8½
Mica, in original cases, medium, lb	3/6 to 6. 0
Petroleum, Russian spot	9¼
Quicksilver, bottle	7.10. 0
Sal-Ammoniac, lump, 1st delivered, U. K., per ton	44. 0. 0
Sulphate of ammonia, f. o. b. Liverpool, ton	13. 0. 0
Sulphur, recovered, ton	5. 5. 0
Shellac, per cwt	4. 5. 0
Platinum, per oz	9. 5. 0
Tin ore, 70%, per ton	142 to 147. 0. 0
Zinc, Vielle Montagne	28.12. 6

Differences on the Month

Higher.	£ s. d.	*Lower.*	£ s. d.
Lead	1.15. 0	Copper	15. 0.
India rubber	4	Tin	13.10. 0
		Haematite	1. 3
		Scotch Pig	6. 9
		Cleveland	6. 9
		Aluminum	5. 0. 0
		Copper sulphate	12. 6
		Sulphate of ammonia	17. 6
		Tin ore	3. 0. 0

Recent Chemical and Metallurgical Patents

Gold and Silver

Precipitation of Cyanide Solutions.—In the method of precipitating cyanide solutions by means of zinc dust, Mr. Charles W. Merrill, of Berkeley, Cal., has discovered that it is desirable to maintain reducing conditions not only to the entrance of the press, but also throughout the precipitation and filtration in the press. Owing to the fact that some solutions filter freely, and that in some cases the supply of solution to the filter is insufficient to keep the press entirely full, the upper portion of the press is apt to contain air or other oxidizing gases. The effect of this air is to oxidize the mixture or the solid component, and thus decrease the efficiency of precipitation and the cost of the precipitant.

In presses which discharge their filtrate from the bottom, it is not always possible to maintain reducing conditions in the press, and Mr. Merrill proposes to adopt a discharge at or near the top of the press, so that the filter will always be full of the mixture, and thereby avoid the accumulation of air in the press. Any filter or press from which the atmosphere is completely excluded and which is capable of being maintained full of the mixture during the operation of filtration, may be used in connection with this process. (1,063,567, June 3, 1913.)

Alloy of Zinc as Precipitant.—The preparation and use of an alloy of zinc, lead, magnesium, aluminium, or other similar combination, is the basis of several patents granted to Mr. Charles W. Merrill, of Berkeley, Cal. The inventor claims that such an alloy is a more efficient precipitant than zinc alone. In order to prepare the alloy in a finely divided state, it is mixed with an abrasive before cooling, whereupon it can be readily reduced to powder by any form of comminuting apparatus. The use of the abrasive is essential to overcome the natural malleability of the metals which ordinarily prevents their comminution.

Another form of application of the precipitant is to make use of it as a liner and balls for a small tube mill, through which the solution passes. The resulting attrition is sufficient to liberate the metal and allow it to flow with the solution to the collecting press. The alloy has a galvanic action which makes it a more efficient precipitant than one metal alone. (1,063,568-69-70, June 3, 1913.)

Zinc, Lead and Copper

Coating Metals with Lead.—While there has been comparatively little difficulty in coating metals with tin and zinc, it has been found almost impracticable to coat such metals with a uniform covering of lead by immersing them in a bath of molten lead. The difficulty lies in properly fluxing the surfaces of the metal at the relatively low temperature of molten lead. According to an invention of Mr. Samuel Edwards, of Pittsburgh, Pa., a suitable flux which will permit of such coating, is composed of the following ingredients: 100 lb. hydrochloric acid in which is dissolved 2½ lb. of zinc, and 100 lb. hydrochloric acid in which is dissolved 1 lb. aluminium and ½ lb. sal ammoniac. The two solutions are mixed. The articles to be coated are immersed in the bath thus formed, and while their surfaces are still wet with the flux, they are dipped in a bath of pure lead heated to a temperature of about 600 deg. F. By the use of this flux it is possible to make as firm and adherent a coating of lead as with zinc or tin. (1,063,248, June 3, 1913.)

Recovery of Zinc and Copper from Brass Waste.—In the ordinary method of melting brass turnings or waste, zinc is largely lost and copper alone recovered. According to an invention of Mr. Harvey M. Burkey, of Newark, N. J., brass turnings or other forms of brass waste are mixed with finely divided fuel, such as anthracite coal, and subjected to an oxidizing blast in a converter of the Huntington-Heberlein type. The temperature and volume of air are so regulated that the zinc is oxidized, volatalized and collected in a suitable system of bags, while the copper remains behind as a sinter which can be charged directly into a reverberatory furnace and refined. (1,061,447, May 13, 1913.)

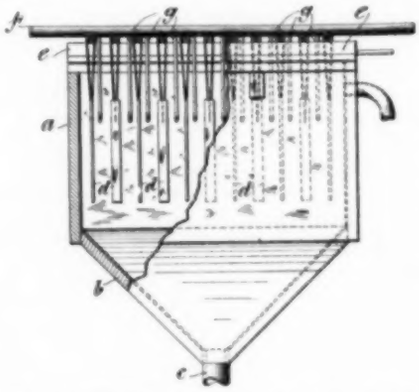


FIG. 1.—METHOD OF CIRCULATING ELECTROLYTE

Roasting Copper Ores.—The conversion of copper oxides or carbonates into sulphates in order to facilitate their treatment by hydrometallurgical methods, is the basis of a patent granted to Mr. Utley Wedge, of Ardmore, Pa. The inventor employs a furnace of the superimposed-hearth type, and mixes copper oxide or carbonate ores with a sulphide such as pyrite. This mixture is treated under such conditions of oxidation and temperature as will convert the copper into sulphate, the iron being first changed into sulphate and then reacting with the copper. In case pyrites is the form of sulphide used, a temperature of 800 to 1000 deg. F. is required. Some sulphur dioxide and trioxide will be formed, and in order to utilize these gases the furnace is designed for a downward draft of evolved gases, so that the material is subjected to the action of the gases from the time it enters until it leaves the furnace. This tends to increase the conversion of copper oxide or carbonate into sulphate. The product is then suitable for leaching and recovery of the copper by any method of precipitation. (1,063,629, June 3, 1913.)

Electrolysis of Copper at High Current Densities.—In the electrolytic refining of copper and the recovery of anode slime, it has been impossible to use currents of high density on account of the uneven deposition of the copper and the consequent development of short-circuits. Agitation or circulation of the electrolyte would avoid the short-circuiting, but would cause

contamination of the cathode with anode slime. In order to overcome these difficulties, and permit electrolysis at high current densities without contaminating the cathode deposit, Mr. KENNETH GUITERMAN, of New York, N. Y., proposes a method of circulating the electrolyte which will continuously remove the liquor to a filter, where the anode slime will be collected, and return a clean electrolyte to the cell. Fig. 1 shows a side elevation of an electrolytic vat, partly in section, with the arrangement of electrodes and feed pipes.

Electrodes are shown at *d*, suspended in the usual manner

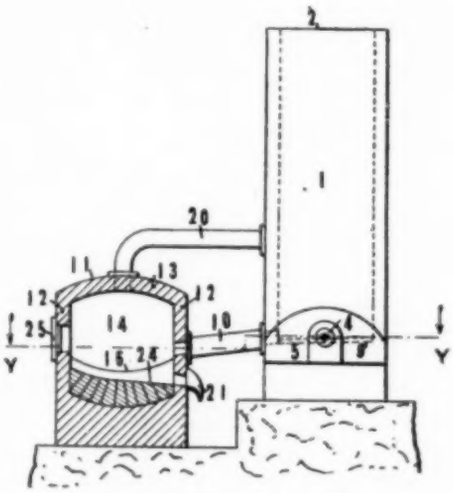


FIG. 2.—FURNACE FOR MELTING AND REFINING COPPER

from bus-bars *e*. The electrolyte is supplied through the pipe *f*, and thence through smaller pipes *g*, the latter being curved at their lower end so as to direct the stream across the electrodes and remove the anode slime. At the same time the electrolyte is constantly flowing out of the vat at *c*, and passing to a filter where the slime is collected, after which the liquor is again returned to the vat through the supply pipes. By means of this process a high rate of circulation is maintained between the electrodes where it is required, so that concentration at the cathode surface is constant. The slime is removed as fast as formed, and a smooth, dense cathode deposit is obtained. The process facilitates the treatment of impure material, and increases the capacity of the plant. (1,062,966, May 27, 1913.)

Melting and Refining Copper.—A process for melting and refining copper has been patented by Mr. DAVID W. BLAIR, of New York, N. Y., consisting in melting the copper in a shaft furnace under oxidizing conditions and then finishing the refining process in a reverberatory furnace which is connected with the vertical furnace. Fig. 2 shows an elevation of the vertical furnace, and a section of the reverberatory. In carrying out the process, the furnace 1 is charged with cathode copper, which

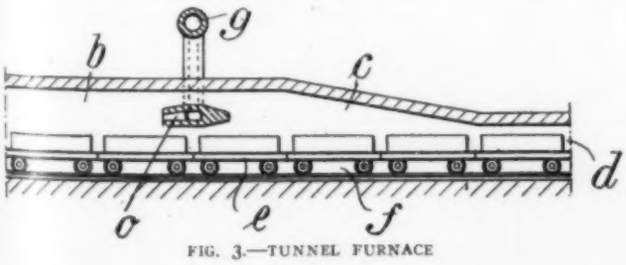


FIG. 3.—TUNNEL FURNACE

melts in the heat generated at the fuel burner 4. Conditions are maintained so that the melted copper delivered to the refining furnace 14 will contain some slightly oxidized copper. A fuel burner situated at one end of the refining furnace maintains the metal in a molten condition. The refining process consists in spreading a reducing agent, such as carbon, over the surface of the bath, and then poling the charge in the usual manner until the oxide of copper is reduced.

By insuring the presence of some oxygen in the copper passing to the refining furnace, a means of controlling the refining process is secured. For example, should the copper in the refining furnace be overpoled, oxidized copper from the melting furnace could be added to correct the condition. The hearth 24, of the refining furnace is formed of wedge-shaped blocks of refractory material, the form of which prevents copper from forcing the blocks out of place, for should any metal find its way beneath the blocks, its pressure would be resisted by the inverted arch structure of the hearth. (1,058,941, April 15, 1913.)

Metallurgical Furnaces

Tunnel Furnace.—An improvement in tunnel furnaces has been patented by Mr. ARTHUR RAMEN, of Olympia, Sweden. The object of the construction shown in Figs. 3 and 4 is to introduce into the combustion chamber the preheated air from the inner end of the cooling chamber, and mix the air with the gas or fuel used in combustion. According to the invention, there is arranged between the cooling chamber *c* and the combustion chamber *b*, a chamber *o*, formed of brick, placed across the furnace, and open to the combustion chamber. Above and beneath the chamber *o*, the combustion and cooling chambers are in communication with each other. Gas is introduced under pressure into the chamber *o* from the conduit *g*, and passes into the combustion chamber between two air currents, thus causing by injector action an abundant supply of air from the inner end of the cooling chamber. At the same time an intimate mixture of air and gas is formed, causing complete combustion and a high temperature. (1,062,606, May 27, 1913.)

Rabble for Mechanical Roaster.—To facilitate the attachment and removal of rabbles from the stirring arms of mechanical roasting furnaces, Mr. MAURICE VAN MARCKE DE LUMMEN, of Cologne, Germany, has designed a special form of stirring device. The lower face of each arm rises obliquely in the direction of rotation, and has an obliquely rising guide for each rabble. The latter, which is loosely inserted in the guide, is thus forced against the under face of the arm by the counter pressure exerted by the roasting material, and is thus safely and firmly secured in position. In order to remove a rabble it is necessary only to withdraw it by means of a hook which can engage with a notch or opening in the rabble. By this arrangement, any rabble can be replaced without disturbing the other rabbles on that arm. (1,061,303, May 13, 1913.)

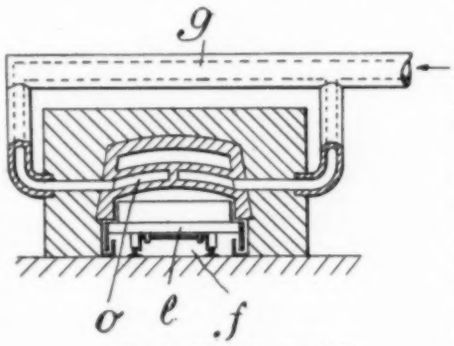


FIG. 4.—TUNNEL FURNACE

Combination Blast Furnace and Converter for Copper.—According to an invention of Mr. HOWARD F. WIERUM, of Upper Montclair, N. J., it is proposed to design a blast furnace so that in addition to the regular production of copper matte, the converting process can be carried on, producing either metallic copper or white metal. To this end the furnace is provided with additional tuyeres for the introduction of air and silica into the matte formed in the crucible of the furnace. The advantages claimed by the inventor are the further oxidation of sulphur and iron, the conservation of the heat produced by this oxidation, and the enrichment of the sulphurous gases given off from the furnace.

One form of the invention is shown in Fig. 5, which embodies a blast furnace of the usual form, with ordinary low-pressure air delivered through pipes 12 and tuyeres 15. For the further oxidation of the matte, additional high-pressure air is provided from a supply pipe 18 and tuyeres 16. The crucible 3 is of refractory material, and has a tap for metal at 4, and a

recess and spout for slag at 5 and 6. The sole plate 7 is supplied with cooling water through pipes 11. Receivers 19 are provided for silica which is injected into the matte through the high-pressure tuyeres.

Assuming the furnace to have been charged in the usual manner, but with less carbonaceous fuel than is customary, and an accumulation of matte in the crucible, the converting tuyeres are used, and the matte subjected to a bessemerizing action. Silicious flux also is introduced through the converting tuyeres to slag off the iron. Flue dust also may be thus introduced beneath the molten charge. The smelting and converting operations may be successive, continuous or simultaneous. (1,063,486, June 3, 1913.)

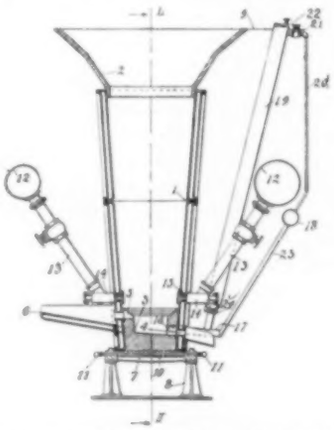


FIG. 5.—COMBINATION BLAST FURNACE AND CONVERTER

Smelting and Refining Furnace.—Another form of furnace designed to effect the operations of smelting and converting in the same receiver, has been patented by Mr. Edward Fink, of Milwaukee, Wis. As shown in Fig. 6, it comprises a rotary roasting or preheating barrel *a*, and a rotary smelting furnace *b*, both mounted and operated in the usual manner. The smelting furnace is further provided with three sets of tuyeres *w*, one set of which is shown. Blast for the tuyeres is supplied by blower *z*, through the wind box *u* and pipes *v*. Fuel is supplied through the pipe *m* at the opposite end of the furnace.

In operation the ore is fed through the preheating or roasting furnace *a*, being discharged into a hopper *k*, and thence fed into the smelting furnace. As the furnace rotates, the tuyeres pass through the molten bath, and air is admitted to the charge. Rapid desulphurization and smelting takes place, and ultimately the barrel becomes filled with slag and matte. When this stage is reached, the furnace is stopped with all tuyeres above the level of the bath, and the matte is allowed to settle. Fuel is supplied through the burner, and the bath kept fluid so that

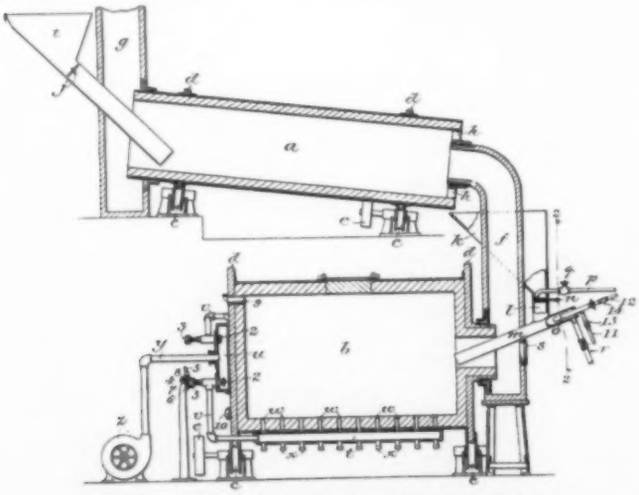


FIG. 6.—COPPER SMELTING AND REFINING FURNACE

settlement of the matte can be complete. The slag is then drawn off through tap holes provided for the purpose at one end of the barrel, and the refining operation begins. During this part of the process the barrel is not rotated, but is rather oscillated so that the tuyeres do not pass through the metal formed. When the matte has been converted into metal, the latter is drawn from the furnace through suitable taps. The slag from the refining operation is used to form a bath for the next charge of ore. In this manner, the same furnace is made to serve for smelting and refining. (1,061,279, May 13, 1913.)

Chemical Engineering

Electrolysis of Black Liquors of Soda-Pulp Mills.—In order to purify the black liquors obtained in pulp mills from the boiling of wood with lye, Mr. Nils A. Langlet, of Gottenborg, Sweden, proposes to use electrolysis in such a manner as to precipitate the ulmic compounds in a form in which they may be easily separated by filtration. One of the conditions necessary to obtain a filterable precipitate is to increase the salt concentration of the liquor. For this purpose salts such as sodium chloride, sulphate or carbonate are added to the liquor, and the precipitation carried out at a temperature of not less than 65 deg. C. The inventor prefers to use a cell with porous partition between the electrode chambers, and to supply the liquor to the anode chamber only, while the cathode chamber is filled with regenerated soda lye or sodium chloride solution. This solution may be used until the ratio between the caustic soda formed in the said chamber and the remaining quantity of sodium chloride is suitable for boiling cellulose material, whereupon it can be used for compensating losses of alkali in the process. The invention differs from others of a similar nature in producing a filterable precipitate. (1,062,016, May 20, 1913.)

Synopsis of Recent Chemical and Metallurgical Literature

Iron and Steel

Microstructure of Sintered Iron-Bearing Materials.—In the May *Bulletin* of the American Institute of Mining Engineers, Mr. B. G. Klugh, of Birdsboro, Pa., presents the results of a study of the microstructure of sintered iron-bearing substances, and compares the value of various processes for sintering such materials. The article is illustrated with microphotographs which show the effect of sintering by different methods. The object of the investigation was to determine the relation of the structure of these materials to their reducibility in the blast furnace. A notable fact demonstrated by the study is that the reducibility of the sintered product is inversely proportional to the degree of fusion which occurs in the sintering process.

Thirteen products were examined, formed of different iron-bearing materials and produced by different sintering methods. Those to which the greatest degree of heat has been applied in sintering show clearly the formation of iron silicate as a glass which coats the particles of sinter and prevents the reducing action of the gases in the blast furnace. Thus the agglomerating process which requires the least fuel for effective binding of the particles is not only the most economical in its own operation, but also produces the ideal material for the blast furnace.

The products examined and illustrated are arranged in the order of the degree of complete fusion to which they were subjected in order to effectually bind their particles together. This arrangement also presents the materials in reverse order of their desirability as a furnace feed, and is as follows: Heating-furnace cinder; puddle cinder; Heberlein agglomerate of flue dust; Heberlein agglomerate from pyrites cinder; rotary kiln nodules from red hematite ore; rotary kiln nodules from flue dust by the process of the American Sintering Co.; flue dust and pyrites cinder by Dwight & Lloyd process; sinter of Cornwall ore by Dwight & Lloyd process; Norwegian magnetic concentrate by Dwight and Lloyd process; magnetic concentrate by Dwight & Lloyd process; granular hematite ore by Dwight & Lloyd process; flue dust and magnetite by Dwight & Lloyd process; flue dust by Dwight and Lloyd process.

From the observations made on the tests the author arrives at the following conclusions:

1. The permeability of the cell wall of a sintered product varies inversely as the degree of fusion to which it has been subjected.

2. In a product of complete fusion, the silica present combines with its equivalent of iron oxide to form a perfect glass, which, from its greater fluidity, envelops and seals the remaining iron oxide from the action of gases.

3. Conversely of the foregoing conclusion, in the product of the lowest degree of fusion, the iron oxide and slag forming materials as a unit are bonded together by incipient fusion, leaving the predominant iron oxide free and vulnerable to the action of the gases in the highest degree attainable in solid products.

The above salient facts show the Dwight & Lloyd products, when properly made, to possess those properties which distinguish them from the products of other sintering processes or agglomerating methods by freedom from those constituents to which scouring action in the blast furnace is attributed.

Gold and Silver.

Weight of Tube Mill Pebble Loads.—The following tabulation has been prepared by MR. W. A. CALDECOTT, and published in the Feb. 1913 issue of the *Journal* of the Chem. Met. & Min. Soc. of S. Africa:

Weight of Load in Tons of Pebbles (at 105 lb. per Cubic Foot).

Pebble Load in 22 ft. Tube Mill:	Internal Diameter of Tube Mill Lining									
	54 in.	55 in.	56 in.	57 in.	58 in.	59. in.	60 in.	61 in.	62 in.	63 in.
12 in. above axis of mill	14.21	14.65	15.10	15.55	16.02	16.48	16.96	17.44	17.92	18.42
11 in. above axis of mill	13.81	14.25	14.69	15.14	15.59	16.05	16.51	16.98	17.46	17.94
10 in. above axis of mill	13.41	13.84	14.27	14.71	15.15	15.60	16.06	16.52	16.99	17.47
9 in. above axis of mill	13.01	13.43	13.85	14.28	14.71	15.16	15.60	16.06	16.52	16.98
8 in. above axis of mill	12.60	13.01	13.42	13.84	14.27	14.70	15.14	15.59	16.04	16.50
7 in. above axis of mill	12.18	12.58	12.99	13.40	13.82	14.24	14.67	15.11	15.56	16.01
6 in. above axis of mill	11.76	12.15	12.55	12.95	13.35	13.78	14.20	14.63	15.07	15.51
5 in. above axis of mill	11.33	11.72	12.11	12.50	12.90	13.31	13.73	14.15	14.58	15.01
4 in. above axis of mill	10.90	11.28	11.66	12.05	12.44	12.84	13.25	13.66	14.08	14.51
3 in. above axis of mill	10.47	10.79	11.21	11.59	11.98	12.37	12.77	13.17	13.59	14.00
2 in. above axis of mill	10.04	10.40	10.76	11.14	11.51	11.90	12.29	12.68	13.09	13.50
1 in. above axis of mill	9.60	9.96	10.31	10.68	11.05	11.42	11.80	12.19	12.59	12.99
Level with axis of mill (*i.e.*, half full)	9.18	9.53	9.88	10.23	10.60	10.96	11.34	11.72	12.11	12.50
1 in. below axis of mill	8.77	9.10	9.45	9.79	10.14	10.51	10.88	11.25	11.63	12.01
2 in. below axis of mill	8.33	8.66	9.00	9.33	9.69	10.03	10.39	10.76	11.13	11.50
3 in. below axis of mill	7.90	8.27	8.55	8.88	9.22	9.56	9.91	10.27	10.63	11.00
4 in. below axis of mill	7.47	7.78	8.10	8.42	8.76	9.09	9.43	9.78	10.14	10.49
5 in. below axis of mill	7.04	7.34	7.65	7.97	8.30	8.62	8.95	9.29	9.64	9.99
6 in. below axis of mill	6.61	6.91	7.21	7.52	7.84	8.15	8.48	8.81	9.15	9.49
7 in. below axis of mill	6.19	6.48	6.77	7.07	5.38	7.69	8.01	8.33	8.66	8.99
8 in. below axis of mill	5.77	6.05	6.34	6.63	6.93	7.23	7.54	7.85	8.18	8.50
9 in. below axis of mill	5.36	5.63	5.91	6.19	6.49	6.77	7.08	7.38	7.70	8.02
10 in. below axis of mill	4.96	5.22	5.49	5.76	6.05	6.33	6.62	6.92	7.23	7.53
11 in. below axis of mill	4.56	4.81	5.07	5.33	5.61	5.88	6.17	6.46	6.76	7.06
12 in. below axis of mill	4.16	4.41	4.66	4.92	5.18	5.45	5.72	6.00	6.30	6.58

The above table is calculated as follows:

Assume R = Internal radius of lined tube mill in inches.

V = Height in inches from top of pebble load to lining.

L = Length of tube mill in feet.

F = Weight in lb. of one cubic foot of pebbles.

Then the general formula for the pebble load in tons is:

$$0.0001\ L\,F\left\{0.1091\ R^2 - 0.0463\ V\sqrt{V(2R - 0.608\,V)}\right\}$$

The pebble load in tons of a tube mill 22 ft. long, and with pebbles at 105 lb. per cubic foot, is:

$$0.0252\,R^2 - 0.0107\,V\sqrt{V(2R - 0.608)}.$$

The above formulae are derived from Molesworth's formula for the area of a segment, namely:

$$\frac{4\,V}{3}\sqrt{(0.626\,V)^2 + C^2},$$

when V is the height of the segment, and C is the semichord.

When the upper surface of the pebbles is below the axis of the tube mill, find the weight of the pebble load whose upper surface would be the same distance above the axis, and deduct this weight from the weight of the pebble load required to entirely fill the tube mill.

Recovery of Black Sand and Floating Minerals.—An article relating to recovery of black sand on amalgamating plates in Rand mills is published by MR. J. M. NEILL, in the March, 1913, issue of the *Journal* of the Chem. Met. & Min. Society of South Africa. The general practice on the Rand is to collect and treat separately all the black sand that happens to adhere to the amalgamated copper plates. This sand consists mainly of iron sulphide containing large quantities of gold, with small quantities of iridium, osmiridium and in some cases platinum. It generally comes down on the plates near the discharge from the mortar boxes, and is seldom found below the top 2 ft. of the plate. In the opinion of the author much more black sand could be collected if provision were made to prevent its escape from the plates. The failure of all the sand to become attached to the plate is due probably to flotation of small particles, by reason of attachment of small bubbles of air, or of films of air which prevent the mineral from overcoming the surface tension of the water and sinking to the plate. In order to accomplish this purpose the author devised a light, flexible, floating apron of canvas and cork, which extends across the plate at a point about half way down and under which the pulp may flow freely. Air bubbles attached to mineral particles are disturbed and liberated and when the surface tension of the water is overcome the particle sinks and is attached to the plate.

The value of the device in effecting an additional recovery of black sand which ordinarily would find its way into the cyanide pulp together with floured mercury is shown by the records of four months' operation with and without the device on the plates. During each period of four months 70,000 tons of ore were treated and all conditions of temperature, alkalinity, etc., were the same. At no time during either period was the mill allowed to run with acid water. The black sand was collected by the usual methods at the regular times of dressing the plates.

The first period, when the device was not used, yielded 1 ton of black sand for each 6188 tons of ore crushed. The value of the black sand actually recovered was 647 oz. fine gold per ton.

The second period, during which the device was used, yielded 1 ton of black sand for each 5000 tons of ore treated and the value actually recovered was 899 oz. fine gold per ton. Subsequently, when the methods of operation were better perfected, values of 1200 oz. to 1500 oz. per ton were recovered.

During the first period the proportion of black sand amalgam in total amalgam was 33.09 per cent; during the second period it rose to 43.41 per cent. The actual increase in mill recovery was 5.88 per cent of the value of the screen sample.

The separate recovery and treatment of this product by amalgamation offers great advantages over allowing it to pass to the cyanide department, as the percentage recovery is higher.

Consolidated Langlaagte Mill, South Africa.—In the *South African Mining Journal* for May 17, 1913, the Consolidated Langlaagte mill is spoken of as "the last word in Rand milling and cyaniding practice." The following data are given for the first three months of this year:

The aperture of the battery screen varied from 0.375 to 0.5 in., and the falling weight of the stamps averaged 1758 lb. The stamp duties were as follows:

	Tons per stamp per 24 hours.
January	18.230
February	19.545
March	20.135

Grading analyses of the final pulp sent to the cyanide plant were:

	+60 per cent	+90 per cent	+200 per cent	—200 per cent
January	0.69	14.33	23.58	61.40
February	0.27	10.04	22.83	66.86
March	0.12	10.33	20.03	69.52

By means of the double cone system of classification, 39.5 per cent of sand and 60.5 per cent of slime were produced. The actual percentages of gold recovered were:

	January per cent	February per cent	March per cent
By amalgamation	71.982	70.474	72.880
From sand	12.118	12.667	12.221
From slime	12.423	13.607	11.502
Total	96.523	96.748	96.603
Value of residue per ton milled, dwt.	0.221	0.219	0.231

A general outline of the process is as follows: The ore is screened over grizzlies, washed in trommels and sorted on belts. The undersize of the trommels passes to the sand pumps and the coarse ore is broken in jaw crushers preliminary to stamping. One hundred stamps of 1900 lb. weight fall through a height of 8½ in. ninety-eight times per minute. The battery pulp is classified in a double set of cones, the underflow of the large cones passing to the tube mills, of which there are ten, 16 ft. 6 in. long by 6 ft. diameter. The overflow of the primary cone is classified into coarse and fine sand products and flows to the pumps for those products. The tube mill discharge is amalgamated on forty stationary plates, each 7 ft. by 5 ft., four to each tube, and finally elevated to the classifying cones for separation into fine and coarse products for cyaniding. The sand is treated by leaching and the slime by intermittent decantation. Precipitation is by zinc shavings.

Zinc

Formation of Zinc Ferrite in Roasting Blende.—The tendency of oxides of aluminium, chromium, calcium and zinc to combine with Fe_2O_3 at high temperatures has been proved due to the function of Fe_2O_3 acting as an acid, forming in the case of zinc, $ZnOFe_2O_3$, zinc ferrite. As the formation of this compound has an effect on processes for extracting zinc from its minerals, a study of the formation of ferrites was undertaken by Mr. G. S. Brooks, of Depue, Ill., who publishes a paper on the subject in the May, 1913, *Bulletin* of the American Institute of Mining Engineers.

In examining with a microscope the many products of roasting kilns it is impossible to find individual grains of zinc ferrite as formed synthetically. A partial ferritization is noticed, however, in which one end or side of a grain seems to be transformed into ferrite. This, however, is not true of Western marmatites, the product of which seems always uniform.

A method of chemical analysis of roasted products to determine the percentage of ferritized zinc is to use a solution which will dissolve unchanged zinc oxide, and leave any zinc that has been altered to ferrite. A suitable solution, and one that gives results that are consistent, contains 200 gr. NH_4Cl, 500 cc. NH_4OH and 750 cc. water. By digesting 0.5 gr. of the sample in 50 cc. of this solution for an hour, the unchanged zinc oxide will be dissolved, and can be estimated in the usual manner.

In considering the formation of zinc ferrites on the hearths of roasting kilns, two distinct ores must be considered: one a mechanical mixture of iron and zinc minerals, and the other a chemical combination of the two. In the former case contact of the grains plays a more important part than in the latter. In the case of mechanical mixtures of blende and marcasite, the author is of the opinion that the temperature of the roasting furnace is insufficient to cause incipient fusion of the two minerals to any great extent. An ore very high in marcasite will be more sticky, and it is likely that the zinc-iron matte thus formed will yield ferrites. In such ores the fine particles may, at the time of rabbling, flash to a temperature sufficiently high to cause fusion.

As to the progress of formation of ferrites during the roast: It is generally agreed that zinc sulphides in roasting form intermediate products such as normal and basic sulphates. It appears probable that the tendency of the oxides of iron and zinc to form ferrites begins with the zinc as sulphate, or with both metals as oxides, or with the desulphurization of any zinc-iron matter present. The author has found that the two roasted oxides of iron and zinc, when mixed in a very fine condition, do form ferrites to an appreciable extent, notwithstanding the observations of some earlier investigators to the contrary. Zinc sulphate and iron oxide, when roasted together in a laboratory assay muffle at a low heat, were found to yield a high percentage of ferrite.

In regard to the composition of ferrites, the author is of the opinion that the kiln products contain zinc ferrite of greatly varying composition, ranging from a normal ferrite at one end of the series to an extremely basic one at the other, so that its formula at all times cannot be accurately written. Furthermore, no definite relation between the iron-zinc ratio of the green ore to this composition has been discovered.

In discussing the surface factor in the formation of ferrites, the author calls attention to the fact that with the perfect molecular contact of the isomorphous iron in the Western or British Columbia blendes, the size of grain has little bearing on the tendency to form ferrites; but that in ordinary ores, and for equal weights, the portion which will pass 60-mesh screen offers such a surface as will tend to form ferrites more easily than will the coarser pieces. In addition to the area of contact, the frequency with which the ore is rabbled will have a decided bearing on the formation of ferrites.

Rise in temperature increases the tendency for the combination of iron and zinc materials, and it has been observed that the higher temperatures give ferrites toward the more basic end of the series.

The foregoing observations seem to indicate:

1. The possibility of some control over ferritization by reducing the contact surfaces.
2. That highly oxidizing atmospheres in the kiln hearths may permit quicker roasting and lower temperatures, both of which lessen the tendency to combine. This, however, must always allow for the exigencies of acid manufacture with its strong SO_2 gas requirements.
3. That frequent rabbling may aid, when not excessive, to the point of grinding ore grains into fines.
4. That the combination as a ferrite on the hearths may occur with the two minerals as oxides, or as a combined matte, or from the reaction of $ZnSO_4$ and Fe_2O_3.
5. That true marmatites yield a high proportion of ferrite under all conditions of commercial roasting.
6. That the formation temperature in practice is somewhat below 1000° C.

The average tenor of zinc and lead sulphide concentrates in different states is shown by the following table, compiled from statistics of the United States Geological Survey for 1912. A similar table for 1911 appeared in this journal, July, 1912, page 423.

	Per Cent. Metal in Crude Ore		Per Cent. Lead in Lead Conc.	Per Cent. Zinc in Zinc Conc.	Value Per Ton	
	Lead	Zinc			Lead Conc.	Zinc Conc.
Oklahoma—						
Miami District	1.7	2.6	79.6	51.8	$54.52	$39.80
Quapaw District	...	1.1	79.5	57.8	50.00	48.36
Missouri—						
Soft ground	0.27	2.14	79.5	58.3	54.43	49.99
Sheet ground	0.36	1.26	79.6	58.6	55.83	50.84
Illinois	0.4	3.2	79.5	35.4	55.38	27.30
Arkansas	0.1	2.7	79.5	49.0	55.90	39.63
Kansas	0.3	1.6	78.3	55.4	52.88	47.30
Wisconsin	0.2	2.9	76.5	38.4	53.57	28.34

The Adaptability of Electric Welding

By George Hills

Among modern money and time-saving processes, welding by electricity and gases are in the front rank and among the several welding processes now in vogue; the electric arc method is now of particular interest, due to its comparative novelty and its wonderful flexibility.

It is but some nine years ago since I installed, to the best of my knowledge, the first electric arc welding system in America.

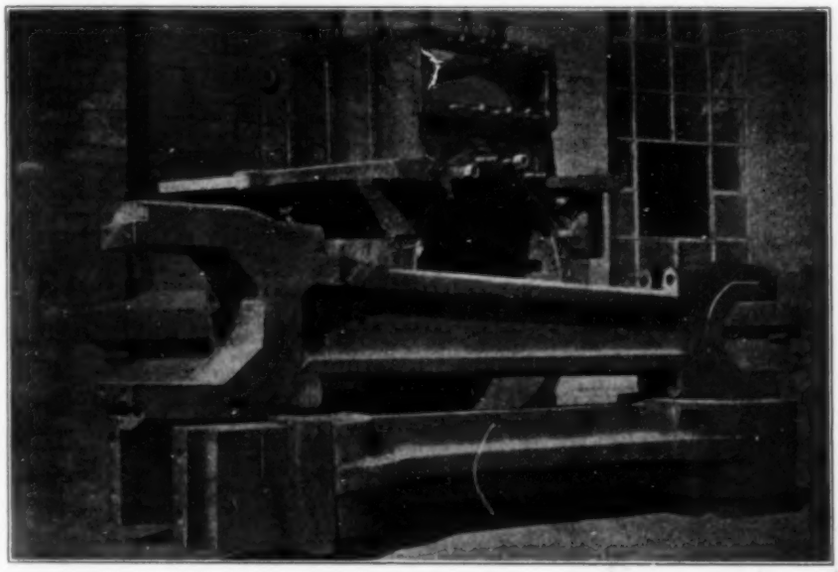


FIG. 1. CONNECTING RODS OF LARGE STATIONARY ENGINE AND CYLINDER WELDED ELECTRICALLY.

This was at the works of the Midvale Steel Company. Scullin-Gallaghers, Commonwealth and other steel companies soon adopted the system for the purpose of patching up and redeeming defective steel castings and forgings, and with what success may be gathered by the fact that each of them trebled their capacity in this direction within a year. Manufacturers of many other lines were quick to see the possibilities of electric welding, and the writer of this article was besieged with questions from all corners of the globe regarding the adaptability of the system.

I have personally engineered the sale of arc welding apparatus during the past thirty months approximating a quarter million dollars and distributed the system amongst steam railroads, electric railways, ship builders, steel foundries, steel mills, tank builders, sheet metal workers, car builders, pipe benders and fabricators, etc. Yet the method is comparatively new, and surprisingly little known. Those who do know the value of electric welding are indeed reaping a rich harvest. Consumers are specifying the welded article in preference to riveted work.

Prices are usually based on gas welding which is from 50 per cent to 75 per cent more expensive than the electric process.

Electric arc welding has been used extensively for the past twenty-five years in England and in Continental Europe for the purpose of uniting iron and steel forming pipe shapes, expansion joints, tanks, reservoirs, galvanizing kettles, etc., and no inconsiderable amount of the product has been imported into this country.

The earliest days of electric arc welding appear to have been about 1881, when this process was employed by one DeMentens for lead burning; a patent was granted to one Bernados in 1887 for iron and steel welding by this method, which was substantially the same as the present day methods although not so economical. The method consists primarily in drawing an arc between the metal to be welded and an electrode, which may consist of a carbon for some classes of work or a metal stick for work of another character, such as flue and locomotive fire-box welding or other work on a vertical wall, or overhead.

In pipe welding either method may be employed with equal success, except where cutting is necessary, for which purpose the carbon electrode is used.

In welding a nozzle to pipe, the hole may be cut to suit, using approximately 400 amperes, the nozzle then fitted and welded with the metallic electrode using approximately 150 amperes. Reinforcement to any desired extent may be added and ofttimes such a fillet is built up to two or three times the thickness of the wall of the pipe.

The metal may be worked and hammered while in a plastic state, thus insuring close grain, equal to drop forgings.

An electric street railway company which has been using electric arc welding on its breakages gives the following interesting data:

Gear case lugs 7 to 10 minutes; electric power 4 to 6 kw.

Armature shafts (broken) 2 inch, 40 to 60 minutes; power 20-30 kw.

Dowel pin holes 5 to 12 minutes; power 4 to 8 kw.

Broken motor cases 2½ to 3½ hours; 75 to 90 kw.

Broken lugs on a compressor cover doors and grease hinges; 2 to 5 minutes, 1 to 3 kw.

Broken truck frames 30 to 60 minutes; the power required varies between 20 and 35 kilowatt.

Worn bolt holes in motors and trucks 5 to 10 minutes; power 3 to 5 kw.

Enlarged and elongated holes in brake levers 2 to 4 minutes; the power required varies from 1½ to 3 kilowatt.

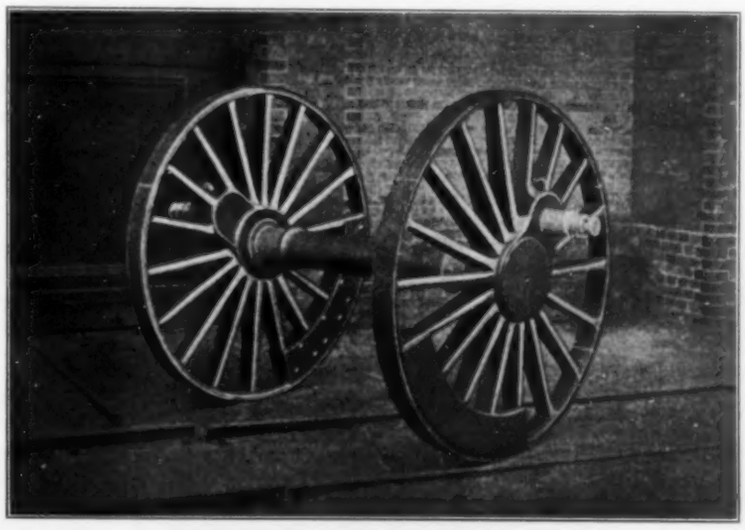


FIG. 2—BROKEN DRIVING WHEEL (SHOWING CRACKS)

Armature shafts, worn in journals 2 inch, 2 to 3 hours; 60 to 90 kw.

Armature shafts, worn in key-ways 10 to 15 minutes; 7 to 12 kw.

Armature shafts, worn thread 20 to 30 minutes; 10 to 15 kw,

Air brake armature shafts (broken) 20 to 30 minutes; 10 to 20 kw.

Leaking axle boxes 5 to 15 minutes; 3 to 7 kw.

The following data is compiled from actual practice and may be taken as average:

Carbon Electrode Cutting and Welding

Size.	Time burning out and preparing.	Time to weld.	Amp.	Volts.	Kw.-hrs.
2" x 2"	3 minutes	10 min.	400	65	5.4
3" x 3"	7 minutes	20 min.	400	65	12
4" x 4"	10 minutes each side.... 1 hr. 15 min. to complete welding and cutting.	25 min.	400	65	34
5" x 5"	15 minutes each side.... 2 hrs. complete welding and cutting.	40 min.	450	65	58½
6" x 6"	18 minutes each side.... 2½ hrs. complete welding and cutting.	40 min.	500	65	81

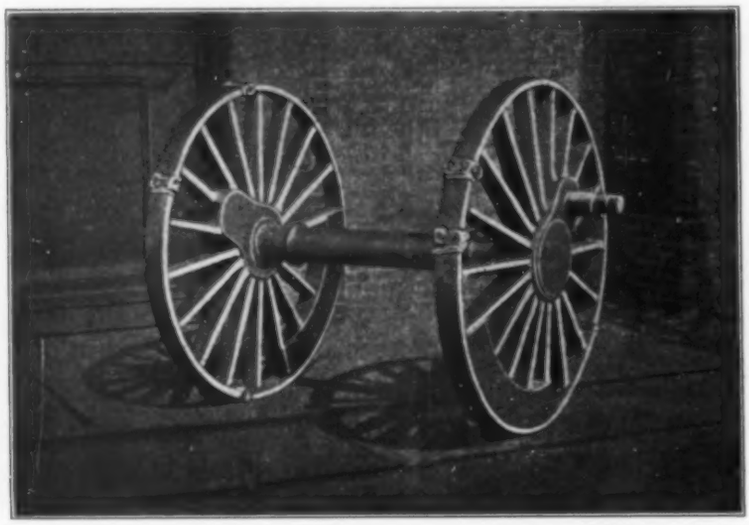


FIG. 3. WELDED DRIVING WHEEL.

Metallic Electrode Welding

Size.	Inches.	Time (minutes).	Amp.	Volts.	Electrodes.	Kw.-hrs.
1/8	6	1 min. 40 sec.....	25	65	1/16	0.04
1/16	6	1 min. 25 sec.....	40	65	3/32	0.06
1/8	4	1 min.	70	65	3/32	0.1
1/4	4	3 min.	100	65	1/8	0.33
3/8	6	12 min. reinforced....	135	65	5/32	1.75
3/8	6	8 min. no reinforcement	100	65	5/32	0.80
1/2	6	12 min. reinforced....	135	65	5/32	1.75
1/2	6	11 min. no reinforcem't	135	65	5/32	1.70
3/4	6	31 min. reinforced.....	140	65	5/32	4.60
3/4	6	25 min. not reinforced.	140	65	5/32	3.65
1"	6	44 min. reinforced.....	140	65	5/32	6.70
1"	6	40 min. not reinforced.	140	65	5/32	6.10

The electric arc method of welding is obviously superior to any method which employs a mixture of carboniferous gas and oxygen, such as oxy-acetylene. In the case of the electric arc the heat is generated by resistance and there is no carbon or oxygen present to combine with the metal at high temperature, whereas, with systems of welding by use of gases, the flame is caused by the liberation of heat for acetylene.

Acetylene is an endothermic compound; that is, its formation is attended by the absorption of or storing up of heat, in contra-distinction to those exothermic bodies which evolve heat in their formation, and the intense heat of the flame is caused by the combining of this gas with oxygen, shown by formula:

$$C_2H_2 + 5O = 2CO_2 + H_2O$$

The adjustment of the proportions of acetylene and oxygen must be very fine, or either the oxygen or acetylene will be in excess. In the former case the superheated steel would be oxidized, while in the latter case it would be carbonized. In either case the strength of the metal would be seriously impaired.

The British Association of Civil Engineers investigated this subject recently, and a paper was read before them by Messrs. T. E. Stanton and J. R. Pannell, and I take the liberty of quoting below from the synopsis of the same, as it bears out the results of my own investigations.

"The strength of welded joints has been investigated by the National Physical Laboratory of Great Britain, at the request of Sir John Wolfe Barry, and the results show that with good workmanship the joint has somewhat better than 80 per cent efficiency, with 1¼-inch bars, the only size tested.

"The experiments were made under rather unusual conditions, for the 167 welded joints received for test were furnished by sixteen firms invited to submit them as samples of the methods of working which they preferred. The bars were subjected to tensile tests and to alternating stresses by the Wohler method. In the latter the machine was run at 2200 r.p.m., after experiments at that speed and at 200 r.p.m. had shown that there was no difference in the fatigue strength of the same material.

"The experiments were described in detail in a paper which Messrs. T. E. Stanton and J. R. Pannell presented on December 12 before the Institution of Civil Engineers. They showed that the mean tensile strength of hand-welded iron bars was 89.3 per cent of the strength of the original bar, and that the percentage in the case of the hand-welded steel was 81.6, with electrically welded iron 89.2, and with electrically welded steel 93.4. Joints made by the oxy-acetylene process were submitted by two makers, but the results obtained with them were not comparable with those obtained by hand or electrically.

Fig. 1 shows connecting rods of large stationary engine and cylinder-welded. Plate 2 shows driving wheel broken. Plate 3 shows driving wheel welded. These are typical repairs.

Butte-Anaconda Electric Railway

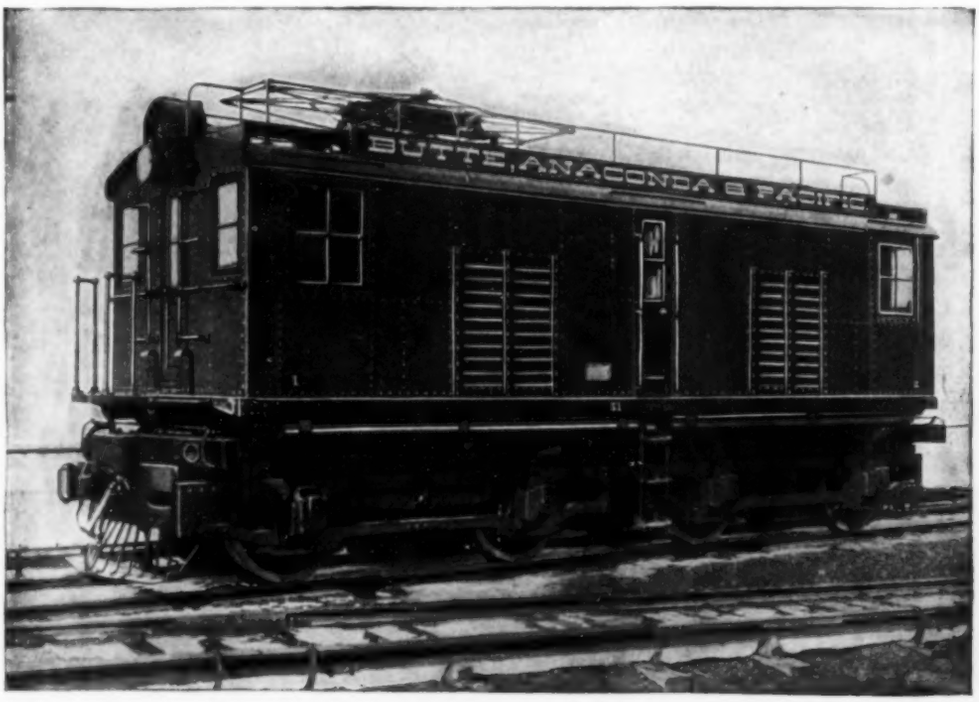
While electric traction problems and the competition between the single-phase and the direct-current systems, important as they are, are somewhat out of the sphere of interest to the readers of this journal, the electrification of the Butte, Anaconda and Pacific Railway is of such exceptional interest as to deserve special mention at this place.

This new electric railway is of interest not only from a metallurgical standpoint, on account of its connecting the Washoe smelter with the Butte mines in a new and very efficient way, but also from a purely electric engineering standpoint, as it is the first installation in this country utilizing the 2400-volt direct-current system. The construction work necessary to effect the change from steam to electric equipment is now practically completed.

The adoption of the 2400-volt direct-current system for this railway was determined after a comprehensive study of local conditions and requirements. The traffic demands are unusually severe and consist principally of hauling long trains conveying copper ore over heavy mountain grades. In comparison with other existing systems, the 2400-volt direct-current system was considered best suited for service of this character.

The Butte-Anaconda section, which has now been equipped, comprises 30 miles of main-line single track and numerous sidings, yards and smelter tracks, aggregating a total of about 90 miles on a single-track basis. The haulage of copper ore from the Butte mines to the smelters at Anaconda, together with all mine supplies, lumber, etc., moving in both directions, amounts to practically 5,000,000 tons of freight per year. Complete freight trains weighing 3400 tons are made up of fifty loaded steel ore cars and will be handled against a ruling grade of 0.3 per cent by a locomotive consisting of two of the units illustrated. Single units will be used for making up trains in the yards and for "spotting" cars.

The initial equipment consists of seventeen locomotive units, fifteen for freight and two for passenger service. Each unit weighs approximately 80 tons. The two units for forming the freight locomotives in each case will be coupled together and operated in multiple unit. The combination freight locomotives will haul the usual trains of 3400 tons at a maximum speed of 15 miles per hour against the ruling grade and at 21 miles per hour on level tangent track.



80-TON 2400-VOLT D. C. LOCOMOTIVE FOR BUTTE-ANACONDA RAILWAY

The passenger locomotives are of the same design as the freight locomotives, but they are geared for a maximum speed of 45 miles per hour on level tangent track. A schedule of eight passenger trains per day, four each way, is maintained, the average train being composed of a locomotive and three standard steam-road passenger coaches.

All the locomotive equipment, as well as the substation apparatus and overhead line material, was designed and built by the General Electric Company.

New Regenerative Heating Furnace

A new regenerative heating furnace, invented by Mr. Friedrich Siemens, of Berlin, Germany, has lately come into extended use in several of the largest rolling mills of Germany and Austria. While it has been described in *Stahl und Eisen*, 1912, No. 37, and in the *Iron and Coal Trade Review* (London) 1912, Nov. 8, it is little known in this country. As the furnace is now to be introduced, however, into this country by Dr. Karl Georg Frank, 90 West Street, New York City, as Mr. Siemens' representative, the following description based on the above mentioned publications should be of interest.

The main point to be kept in view when designing a heating furnace is not so much to develop inside the furnace all the heat contained in the fuel, but rather to transmit to the material to be heated the maximum possible amount of heat, even though part of the total heat is intentionally allowed to go to waste.

The transmission of heat depends, as is well known, on three factors—conduction, convection and radiation. The first two of these increase in approximately direct proportion to the difference in temperature, whereas—according to the Stephen-Bolzmann formula—radiation increases as the fourth power of the absolute temperature. Hence, radiation will finally predominate to such an extent that one can assume, with sufficient accuracy for practical furnace work, that, at high temperatures, the transmission of heat is equivalent to the fourth power of the absolute difference in temperature. This indicates the necessity for a flame of the highest possible temperature in order to work economically, and this flame is best obtained by gas firing with preheating of gas and air.

In the correct assumption that the application of the regenerative system to the heating furnace would mean economic advantage, a number of practical experiments were made, none of which, however, led to the desired result or secured the wished-for advantage over the recuperative furnace, and the claimed advantage over the recuperative furnace, which naturally was the strongest competitor of the regenerative furnace, was not proved.

The types tried were constructed on the principle of reversing the flow of the combustion gases issuing at the charging end of the furnace, and diverting them alternately into heat accumulators. The reasons for the non-success of this type are manifest. Since the flame or used gas was obliged in part to flow over cold ingots, the temperature was lowered too far to enable the accumulators to be heated strongly; and furthermore, the highest temperature in the accumulators was at the point opposite to the air exit, so that, in order to maintain the counterflow principle, it was necessary to construct long intermediate passages.

Finally, a valve capable of diverting the hot gases was required; and it has been found impossible up to the present to overcome, anything like completely, the practical difficulties presented by such valves, exposed as they are to great heat. This type of furnace is, in principle, nothing more than the double-reversing system, of which so much was heard in the early days of the regenerative system, at the time when the change of direction of the flame was still regarded as a drawback.

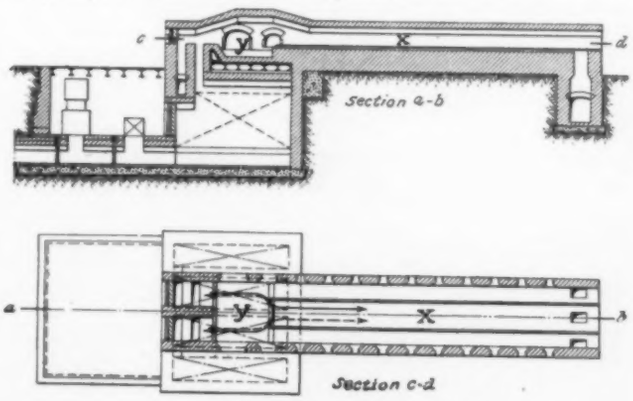


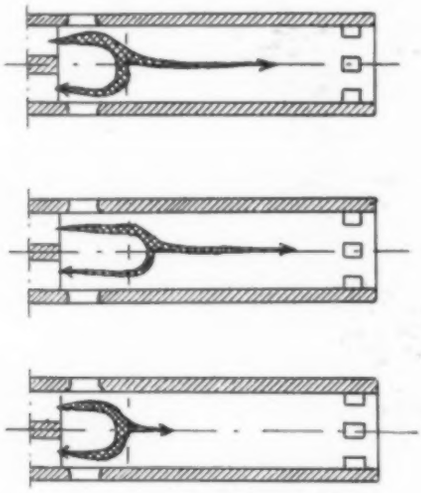
FIG. 1.—REGENERATIVE GAS FURNACE

An entirely novel solution of the problem is afforded by the heating furnace illustrated in Fig. 1. In this type the flame is divided in the open chamber of the furnace, the one portion flowing right through the heating chamber, while the other turns back, horseshoe fashion, and escapes through ports situ-

ated close to the admission ports. After reversal, the disposition of the flame is just the same as described.

At first it was frequently objected that the flame would not take the prescribed course; and this objection is tenable so long as a regenerative flame is not in question, because a flame whose molecules are not highly heated and therefore does not contain any store of energy in itself, must necessarily terminate in a chaos of soot directly the mechanical intermingling ceases at the point where the flame is to divide, and where, therefore, for a short time there are no differences of pressure.

The highly-heated regenerative gas flame, on the contrary,



FIGS. 2, 3, 4.—ADJUSTMENT OF FLAME

overcomes this dead point with ease; the flame does not lose its heat for a moment at the dividing point, and the two divisions burn clearly and perfectly. The reversible part of the flame therefore comes in contact solely with the hot charge, and is thus well adapted to strongly heat the regenerative chambers; or, in other words, these chambers can be raised to the maximum temperature with a comparatively small amount of flame.

Like all regenerative furnaces, these work with natural indraft of the air and with an outward pressure in the heating chambers, so that no air can enter when the discharging door is opened. Fig. 1 shows a furnace of this kind in longitudinal and cross section, *x* being the delivery or welding portion and *y* the preheating of "push" portion of the bed.

This furnace is intended to be operated with cold producer gas or blast-furnace gas, and is consequently provided with four adjacently situated regenerative chambers. The arrangement of the air and gas valves is the same as in the ordinary Siemens furnace, except that the flame ports are not situated at both ends of the furnace, but adjacently at the delivery end.

The flame, therefore, travels for a time in the direction of the plain-line arrow, and after reversal, in the direction of the dotted arrow. This is the usual arrangement; but there is nothing to prevent the horseshoe portion of the flame from being assigned a vertical position, and similarly the flame may be led over the welding hearth in a direction parallel to, but alternately in opposite direction to that portion of the flame which flows through the "pushing" hearth *y*.

It is also possible to arrange for a transverse flame, the one portion of which shall be diverted to the "pushing" hearth. There is a further modification with a large central burner and two smaller ones on the right and left of same, both these lateral burners leading to the common heat accumulator. This pattern works with great success.

For metallurgical "push" furnaces, however, the arrangement of the flame as in Fig. 1, with cold or blast-furnace gas, will undoubtedly remain the most usual type.

Accurate measurements and heat balances for these furnaces are now being compiled; but it has already been ascertained that, given proper adjustment of the valves and dampers, the temperature of the escaping gases both at the charging end and the heat-accumulator end of the furnace is about 300 deg. C., and that the mean temperature of the air of combustion is very near 1100 deg. C.

The "push" furnaces previously in use do not in the least fulfil the divergent requirements which must be imposed upon them in respect of controllability. The only method of control hitherto available was the imperfect one of admitting an insufficient, correct, or excessive amount of air, in order to be able to adapt the furnace in any way to meet the various requirements existing when hard or soft, cold or warm, thick ingots or thin slabs are to be heated.

If large pieces of soft steel are in question, the heat on the welding hearth, *x*, may be raised to the highest welding temperature without objections, whereas it is not desired to bring the "push" hearth *y* too quickly to a temperature that will cause an appreciable loss of metal. In these circumstances, with the new regenerative "push" furnace, the flame would be adjusted as shown in Fig. 2, and the thickness of the indicating (arrow) line approximately corresponds to that of the flame and its radiation.

If hard steel is to be heated, then an accurately regulated temperature must be provided on the welding hearth, *x*, together with gradual uniform preheating. In such case the course of the flame should be approximately in accordance with Fig. 3.

If the material is charged warm, it is true that a high temperature will be required on the welding hearth, *x*, but only a gentle heating up on the "push" hearth *y*, since otherwise there would be an unnecessary waste of metal. In this case the regulating valves and dampers must be adjusted to give a flame as in Fig. 4.

In this way it is possible to insure a uniform output, in one and the same furnace, and under the most favorable conditions, no matter whether the charge be entered cold or warm. When thin billets or slabs, which heat up quickly, are treated, the possibility of obtaining a very high temperature on the welding hearth *x* will be utilized, and the furnace made as short as possible, in order to save room and reduce the loss of metal. In this and similar ways, the distribution of heat and the tem-

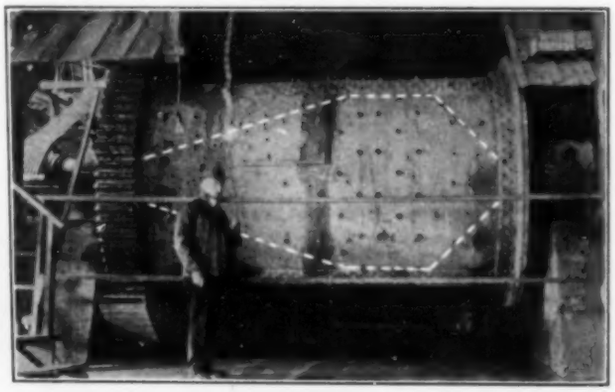


FIG. 1.—TUBE MILL CHANGED INTO CONICAL MILL

peratures can be accurately adjusted, with this furnace, to the requirements of the moment.

This regenerative gas-fired "push" furnace fulfils modern requirements in so far as the present practice is to heat up ingots, blooms, or billets to higher temperatures than was formerly the case, and because of the tendency in large steel works to make use of blast-furnace gas, either alone or in conjunction with producer gas or coke-oven gas, in the rolling-mill furnace.

In the steel works of Germany and Austria alone, 42 of the above-described "push" furnaces have been started or are in course of erection during the last two years, their aggregate capacity being three million tons per annum.

Conversion of a Tube Mill into a Conical Mill

The adjoining illustrations of the conversion of a cylindrical tube mill into a Hardinge conical mill at the plant of the Federal Mining and Smelting Company, at Wallace, Idaho, is interesting in more than one respect.

This company had six Huntington mills in operation, but decided to install some other kind of mill and for this purpose started a preliminary competition. This competition was be-

FIG. 2.—VIEW OF LINING

tween a 7 ft. x 12 ft. cylindrical tube mill and an 8-ft. Hardinge conical mill. The large power consumption of the cylindrical tube mill as well as the impossibility of starting it under full load with a 100-hp. motor resulted in the desire to convert it into a Hardinge conical mill by inserting a special conical lining. As such a conversion constituted a direct infringement of the Hardinge patents, the Hardinge Conical Mill Company furnished the user with a special license to continue the use of the mill as reconstructed.

Fig. 1 (p. 422) shows the outside of this tube mill and the dotted lines indicate the conical lining. This mill is now putting through a little less tonnage with approximately twice the horsepower than the standard Hardinge conical mills.

Fig. 2 is a photograph of the lining. It is of particular interest to note that when Mr. Hardinge invented and developed the conical mill, the first big mill he actually built was such a

FIG. 3.—TWO CONICAL MILLS IN OPERATION

tube mill provided with a conical lining.

Fig. 3 shows two of the six standard 8-ft. Hardinge conical mills which have been installed by the Federal Mining and Smelting Company after the preliminary competition and which are now operating in conjunction with the converted tube mill.

Personal

Mr. **Gelasio Caetani,** metallurgical engineer of San Francisco, was in Denver recently on his return from a trip abroad.

Mr. **Pedro P. Calvo,** of Bogota, is at present in New York to study recent chemical and electrochemical developments with a view of introducing some of them in the Republic of Columbia, which is rich in waterpowers and natural resources.

Mr. **Edwin M. Chance,** for four years chemist of the Philadelphia and Reading Coal and Iron Company, has resigned his position to become consulting chemist to a number of the anthracite and bituminous coal companies, and is now located at 61 South Pennsylvania Avenue, Wilkes-Barre, Pa.

Dr. **Regis Chauvenet** was recently elected president emeritus of the Colorado School of Mines, and Mr. **W. G. Haldane** was appointed acting president. Dr. Chauvenet was president of the school about ten years ago, and Mr. Haldane has been assistant professor of metallurgy. Other changes made in the faculty included the appointment of Mr. **Harry J. Wolf** as professor of mining.

Mr. **John A. Davis,** of the Denver branch of the Bureau of Mines, is engaged in an extensive study of western milling methods, and will be in the field for several months.

Prof. **Stephen L. Goodale,** of the department of ore dressing and metallurgy, University of Pittsburgh, has been visiting in Colorado.

Mr. **Franklin Guiterman,** of the American Smelting & Refining Co., New York, made an inspection of the company's Colorado smelters in June.

Mr. **H. W. Hardinge,** having just returned to New York from an extended trip through the Northwest and British Columbia, is to sail shortly for a two or three months trip to Russia and other countries.

Dr. **J. W. Holmes,** director of the Bureau of Mines, was a guest at a luncheon of the American Mining Congress in Denver in June.

Dr. **Charles L. Parsons,** of the United States Bureau of Mines, has completed an investigation of the uranium and vanadium deposits of southwestern Colorado and southeastern Utah. He was accompanied on the trip by Mr. K. L. Kithil of the Denver office of the Bureau.

Mr. **Edward S. Wiard,** metallurgical engineer of Denver, Colo., is engaged in testing ores for the Vindicator Consolidated Mining Co., Cripple Creek, and designing a plant for their concentration.

Prof. **Paul Walden,** of Riga, Russia, is the subject of a brief biographical sketch, written by Dr. G. F. Kunz, under the title "The President of the Ninth International Congress of Applied Chemistry," in the June issue of the *Popular Science Monthly.*

Digest of Electrochemical U. S. Patents

PRIOR TO 1903.

Arranged according to subject-matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Concluded*).

Refining Metals

312,814, Feb. 24, 1885, Henry R. Cassel, of New York, N. Y.

Relates to apparatus and process for electrolytically refining metals. The apparatus comprises a decomposition tank, a silver precipitation tank and a device to circulate the electrolyte. The anode consists of the metal to be refined and is contained in a bag of horse-hair or asbestos; within the bag is a frame of metal not attacked by the electrolysis. The cathode is surrounded with a bag of a dense, porous, nonconducting substance which permits the passage of the electric current and of hydrogen, by dialysis, but does not permit the passage of dissolved metals from the anode. On passing the current copper and silver in the anode dissolve; any gold present collects as an insoluble residue in the anode bag. Any soluble particles falling from the anode collect on the metal frame and are there dissolved. The acid copper sulfate solution filters through the anode bag into the outer cell where it is precipitated by metallic copper. The solution then runs

off into the precipitating-tank where any remaining silver is precipitated by metallic copper, the solution then pumped into a supply tank from which it flows to the cathode bag or compartment where the copper is deposited and the remaining acid solution overflows to the decomposition tank.

315,265, April 7, 1885, Moses G. Farmer, of New York, N. Y.

Relates to apparatus for refining copper by electrolysis and consists in arranging the deposition vats in rows with spaces between the rows; tracks are placed in the spaces, and a car moved over the tracks to remove electrodes and to pump the electrolyte from a vat to an empty tank for cleaning, etc.

322,169, July 14, 1885, Moses G. Farmer, of New York, N. Y.

Relates to apparatus for refining copper, and consists of a vat in which the ingots are placed so as to divide the cell into a plurality of compartments by fitting tightly between the sides. The metal is dissolved from one side of one plate and deposited upon the adjacent side of the next plate. Against the cathode side of the plates or ingots is placed a grid or latticework of wood, etc., to divide the deposited copper into definite sized units, while the anode side of the plate is being dissolved away. When the operation is completed the cathode may be readily divided into sections along the division lines.

322,170, July 14, 1885, Moses G. Farmer, New York, N. Y.

Relates to a process of refining copper and contains a reference to his companion application which became patent 322,169. The impure copper in the form of plates is placed in inclined grooves in the sides of a vat, the bottoms of the plates resting on transverse bars affording space for the accumulation of the impurities in the metal. The plates so disposed form a plurality of compartments, and are so connected that the underside of a plate is an anode, while the upper side of the facing plate is the cathode. Air pipes supply air to the solution in between the plates, keeping the solution in circulation and assisting in removing the impurities. By this process one side of the plate is gradually dissolved off, while the other side is gradually built up with pure copper.

377,487, Feb. 7, 1888, Edward S. Hayden, of Waterbury, Conn.

Relates to a process for refining crude copper. The metal in plates of suitable size is fitted in grooves in the sides of a vat, which do not reach to the bottom, leaving a space below for the accumulation of impurities. The plates are so arranged that one side is the anode and the other side the cathode, so that a plate of impure metal will be dissolved on one side while pure metal is being deposited on its other side. The impure metal is preferably rolled before refining. The electrolyte is gently circulated through the cell, so as not to disturb the impurities which collect at the bottom.

Book Reviews

Electric Furnaces in the Iron and Steel Industry. By **W. Rodenhauser** and **I. Schoenawa.** Translated from the second German edition, with additions, by **C. H. Vom Baur,** E.E. 5¾ x 9 in. (15 x 23 cm.); 420 pages; 133 illustrations. Price $3.50. New York: John Wiley & Sons. London: Chapman & Hall, Limited.

The first German edition of this work was reviewed in our issue of April, 1911. Mr. Vom Baur has made his translation from advance sheets of the second German edition, and has added here and there such items as were particularly needed by the English and American reader. The record of 90 arc furnaces and 40 induction furnaces in operation or under construction will come as a rude shock to those purblind American steel makers who seem to regard electric steel furnaces as an expensive fad, who do not see their possibilities, and who think eventually to save money by letting them alone. The old, old story is repeating itself, and American industries are again trailing after the European—five years or so after them. The joke would not be so hard on us, if we did not usually pretend or profess to be leading the world. We may be leading the world in making profits, but in general this is not by virtue of our enterprise and superlatively good practice, but simply by reason of our abundant natural resources and high tariff, and in spite of very evident moss-backed conservatism and frequent lack of initiative in adopting technical improvements.

To return to the book, Mr. Vom Baur has made an excellent translation and some useful additions. In particular, the chapter on electric furnace ore reduction to pig iron has been brought "up to the minute," with extensive assistance from Prof. D. A. Lyon. The typographic arrangement is not clearly defined as regards headings and sub-headings; some improvement in this regard should be made in the next edition. Everybody interested in the electric manufacture of pig iron and steel will need this book, for it is the best source of information on that subject now in print.

Handbuch der Mineralchemie. With contributions from fifty-nine special contributors and with the support of the Academy of Sciences of Vienna, edited by Hofat Dr. **C. Doelter,** professor of the University of Vienna. To be complete in four volumes. First volume in 6 parts: vol. I, part 1, (p. 1 to 160), marks 6.50; vol. I, part 2 (p. 161 to 320), marks 6.50; vol. I, part 3 (p. 321 to 480), marks 6.50; vol. I, part 4 (p. 481 to 640), marks 6.50; vol. I, part 5 (p. 641 to 800), marks 6.50; vol. I, part 6 (p. 801 to 1008 and index and title page), marks 9.10. With numerous illustrations, tables and diagrams. Dresden and Leipzig: Theodor Steinkopff.

The first volume of this great new German work is now complete. The parts of the next volumes are following each other in regular rapid succession. The whole work which is to comprise four volumes is expected to be complete in 1914. If from the contents of the first volume, we may by extrapolation draw a conclusion as to the value of the whole work, there seems to be little doubt that this work will become classical in its field.

"In the beginning of its evolution, mineralogy was developed mainly by chemists. But when mineralogy grew to become an independent science and great successes were obtained by physico-mathematical methods, this school became preponderant and the intimate relation between mineralogy and chemistry got lost. Partially this relation was reestablished by the cultivation of mineral synthesis by chemists and by E. Mitscherlich's discovery of isomorphism. . . . Quite recently the application of the principles of physical chemistry to problems of mineralogy has again produced a more intimate connection of mineralogy with chemistry and has opened new and important vistas. . . . This handbook is intended to connect more closely chemistry and mineralogy and to summarize all that chemistry has found out concerning minerals so that the mineralogist as well as the chemist may familiarize himself with anything that has been done in the field common to both."

It may be said right here that the work is really of much broader interest and appeals to many readers outside of the purely mineralogical and chemical fields. Many metallurgical and chemical engineers should find it a source of instruction and inspiration. To mention a single example from the first volume, C. Doelter's extended chapters on silicates and J. H. L. Vogt's concise and illuminating chapter on slags should be of great interest and value to metallurgical engineers.

That fifty-nine specialists, besides the editor-in-chief, have united to write this work, makes its strength. The field is now too big to be covered authoritatively by a single man. Great credit is due to Dr. Doelter, however, for uniting the contributions of his fifty-nine collaborators into a harmonious whole. In this difficult task he has been remarkably successful.

The contents of the first volume are as follows: General introduction. Carbon: diamond, graphite. Carbonates (general, sodium carbonates, magnesium carbonate, calcium carbonate, and other carbonates), carbides, silicon, silicates, glass, glazes, enamels, slags.

The work is undoubtedly at present the most complete and exact summary of our knowledge of the subject and should be in every private or public library which claims in any way to be up-to-date.